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MINING METHODS AND PRACTICES AT THE UNITED VERDE COPPER MINE, JEROME, ARIZONA

BY

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MINING METHODS AND PRACTICES AT THE UNITED VERDE COPPER MINE, JEROME, ARIZ.[1]

By T. W. Quayle[2]

INTRODUCTION

This paper describing mining practices at the United Verde copper mine, Jerome, Ariz., is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

At present mining operations are being conducted both underground and by electric shovels in an open pit. The open-pit work has been described by E. M. J. Alenius in a previous publication of the Bureau of Mines;[3] the present paper will be confined to a description of the underground mine. The underground methods comprise horizontal cut-and-fill, shrinkage with delayed filling, horizontal square-set, inclined square-set, and top slice. The normal daily production of the underground mine is 3,000 tons of copper ore per day, requiring the employment of 900 men underground.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance given by W. W. Lynch, formerly general mine superintendent, Carl E. Mills, assistant mine superintendent, M. G. Hansen, chief mine geologist, W. P. Goss, chief bonus engineer, O. A. Glaeser, ventilation engineer, J. F. Cowley, planning engineer, and others of the United Verde staff in the preparation of the subject matter and illustrations in this paper.

HISTORY

The United Verde mine is sutuated at Jerome, Ariz., on the eastern slope of the Black Hills, at an elevation of approximately 5,500 feet. The town of Jerome is served by the Verde Tunnel & Smelter Railroad, a subsidiary of the United Verde Copper Co., which joins a branch of the Santa Fe Railroad at Clarkdale, 4 miles distant in an air line from Jerome. At Clarkdale is the United Verde smelter.

The original claims covering the United Verde outcrop were located in 1876 by M. A. Ruffner. The outcrop contained oxides of copper and a considerable amount of gold and silver. However, as the terminal of the Santa Fe at that time was in Kansas and the nearest smelter was at San Francisco, the outlook for profit was dark. The United Verde Copper Co. was organized in 1882 and purchased the original claims from Ruffner for $45,000. James A.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6440."

2 - One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, United Verde Copper Co.

3 - Alenius, E. M. J., Methods and Costs of Stripping and Mining at the United Verde Open Pit Mine, Jerome, Ariz.: Information Circular 6248, U. S. Bureau of Mines, February, 1930, 33 pp.

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McDonald and Eugene Jerome of New York were made, president and secretary, respectively. A small reduction plant was installed and shipments of high-grade gold-silver-copper matte began. Financial and operating difficulties, however, caused the operation of the mine and plant to be spasmodic for several years.

Senator W. A. Clark of Montana made his first visit to the mine in 1888, having been impressed by an exhibit of its ores at the World Exposition at New Orleans in 1885. After personally examining the property he secured an option and took control in 1889. Under his guidance developments were rapid and the United Verde eventually became one of the world's leading copper mines.

Senator Clark remained president of the company until his death in 1925. He was succeeded by his son, Charles W. Clark, who now heads the company.

GEOLOGY AND ORE DEPOSITS

The ore deposits of the United Verde mine are of the massive sulphide, schist-replacement type. A structural anticline formed by the intrusion of diorite between bedding planes of steeply-dipping and severely-folded greenstones and porphyry localized the deposition of sulphide. The mineralized zone is of great horizontal extent, varying from 200,000 to 400,000 square feet on various levels, and has a proved depth of 4,000 feet. The irregular inter-fingering of the sulphide with the schists and porphyry, typical of replacement types of ore deposits, forms an irregular and indefinite footwall boundary. The chief primary copper mineral is chalcopyrite; the gangue consists principally of pyrite, quartz, and chlorite.

General Geology

The geologic record begins in early pre-Cambrian times, when a greenstone complex consisting of tuffs, flows, and fragmentals was formed. Younger than the greenstones is a series of siliceous bedded sediments. After the deposition of the latter the formations were folded and tilted steeply to the northwest. Following this the area was intruded by rhyolitic quartz porphyry, which was shortly thereafter rendered schistose by further deformation. Masses of augite diorite then intruded the quartz porphyry and greenstones. The sulphide ore deposits were formed after the intrusion of the diorite. After the formation of the large schist replacement ore bodies, but before the mineralization was entirely completed, a series of small andesite dikes, locally called "water courses," cut the ore masses and other formations. These dikes have an east-west trend which is nearly at right angles to the schistosity of the quartz porphyry. They vary in thickness from a few inches to 50 feet, with an average of 2-1/2 feet.

Continued deformation following the intrusion of the dikes brecciated and displaced all of the older formations. Normal faulting on a major scale caused a vertical displacement of about 2,300 feet on the Verde fault. This fault strikes north-northwest south-southeast. It cut the chimney of sulphide into two segments. A long period of erosion followed, during which the country was base-leveled. It was later covered by the Cambrian seas, which deposited the Tapeats sandstone 0 to 100 feet in thickness.

Overlying the basal sandstones are 300 to 500 feet of Devonian limestone, 300 to 500 feet of Mississippian limestone, and 0 to 500 feet of red sandstone of Permian age. Each of the four periods of deposition was preceded and followed by periods of uplift and erosion.

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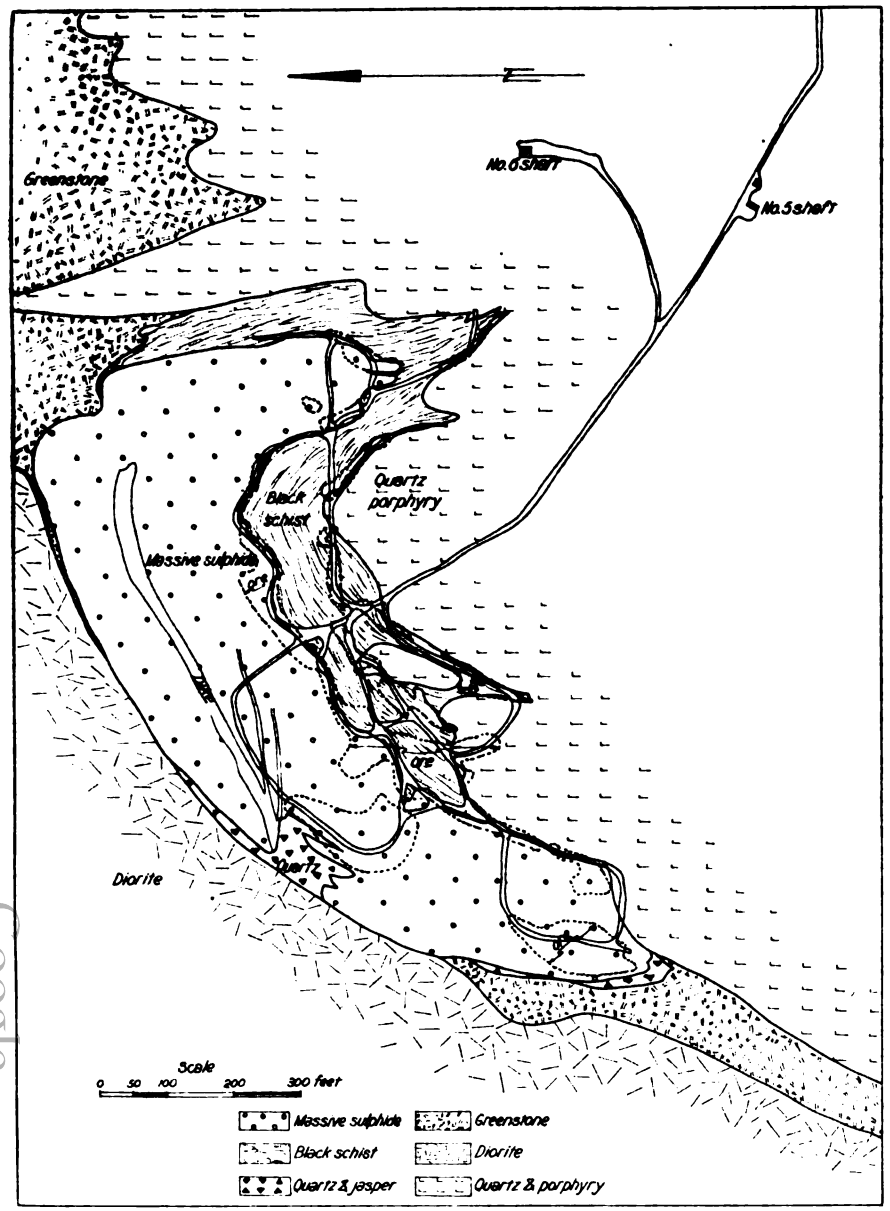

Figure 1.-Geologic map of typical level

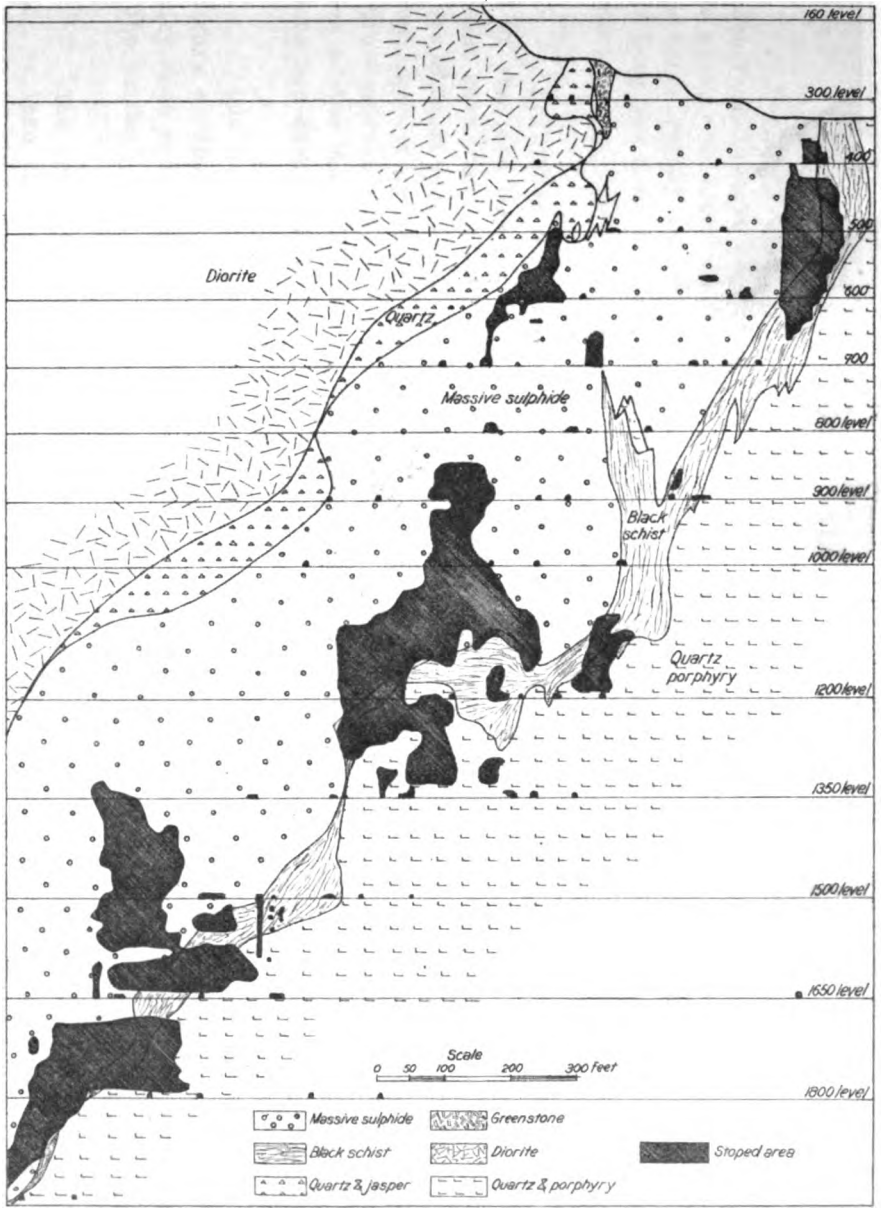

Figure 2.-Geologic vertical section

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Tertiary basaltic lavas are the next recorded formations. These came chiefly from the San Francisco Mountains volcanic field, augmented somewhat by feeders in the form of dikes which are occasionally found in the district. Where the lavas flowed across the Tertiary Verde Valley below Camp Verde, a dam was formed which resulted in a lake at least 35 miles long and 6 or 8 miles wide. In this lake bed (Verde) formations were deposited to a thickness of at least 1,200 feet.

Following the outpourings of lava and more or less contemporaneous with the deposition of the lake bed formations came another period of normal faulting. Several breaks are apparent with a north-northwest and south-southeast trend, also a few attendant cross faults. The Verde fault moved again, causing an additional vertical displacement of about 1,580 feet. The uplifted scarp of the Verde fault was especially subject to erosion so that a strip of the pre-Cambrian was exposed. The erosion on the west side of this fault uncovered the ore bodies of the United Verde mine, whereas on the east side of the fault the pre-Cambrian formations are deeply buried.

Due to the rapid erosion of the steep fault scarp and the ore bodies, very little secondary enrichment exists. Erosion has followed enrichment so closely that the oxidized capping does not extend over 100 feet in depth, and secondary enrichment rarely 200 feet below the oxides.

The ore deposits are of the schist-replacement type. Where the massive sulphides adjoin schists there is a gradual transition from unreplaced to completely replaced schist. The contact is rarely a clean-cut one, and shows all the embayments and irregularities characteristic of limestone replacement deposits. Fingers, pendants, and small isolated patches of ore frequently exist. The ore bodies generally occur along the sulphide-schist contact, and to a less extent within the sulphide, schist, or porphyry areas (see figs. 1 and 2).

The sulphide varies in horizontal section from an elliptical to a roughly crescent-shaped mass with a length of about 1,100 feet along the strike and an average width of about 300 feet. It lies between the chloritic schist footwall and the diorite. Toward the diorite the sulphide becomes hard and siliceous, with practically solid jasper at the diorite contact. The diorite is fresh and hard, with a fairly uniform texture and without much evidence of alteration. The black schists consist essentially of chlorite. The quartz porphyry as a rule is fresh and quite hard, though in places in zones of shearing it may become very schistose and grade into black schist. Occasionally, near the sulphide body where acid leaching has had its effect, the porphyry is altered, softened, and kaolinized.

Physical Characteristics of Ore and Enclosing Rocks

As explained under the heading "Geology," the main ore bodies occur along the contact of the massive sulphide and the schist, with smaller and less important ones on the schist-porphyry contact or entirely in any one of the three rocks. The physical characteristics of the three rocks are so entirely different that the location of an ore body with respect to these rocks determines not only the mining method, but also the size of the stope than can be worked.

The massive sulphide is generally very hard and dense, and stands well over large areas without timber support, except for an occasional bulkhead under a loose slab or where weakened by an andesite dike. This permits mining by cut-and-fill and shrinkage methods

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The schist varies so much in hardness and strength in different parts of the mine that no definite mining method can be assigned to the ore bodies in this rock. At places it stands well enough to be mined by the cut-and-fill method, whereas at other places it requires mining by small square-set sections. Shrinkage is never used in the schist because of dilution from the walls in drawing the broken ore.

In hardness and strength the porphyry forms an intermediate class between the massive sulphides and the schist, and the ore bodies in it can ordinarily be mined by the cut-and-fill method, except in a few places where the porphyry tends to break down in large blocks, leaving the back of the stope so ragged that timbering is required.

There is no general change in the characteristics of any one of the three rocks from the surface to the 3000 level, the present bottom of the mine.

In the massive sulphide and porphyry ore bodies the values shade off from commercial ore to waste, and the stoping limits are determined by assay. This may be true of the schist ore bodies, but more often the change from commercial ore to barren waste is sudden.

METHODS OF PROSPECTING AND EXPLORATION

Ore bodies may be found within an extremely large area, and the so-called "main ore zone" consists of numerous irregular lenses of ore, large and small. Therefore, various methods of prospecting are used, depending upon the area to be prospected and its location geographically and geologically. Diamond drilling is the chief method, followed by drifts, crosscuts, and raises.

The preliminary work in new and undeveloped country is done by diamond drilling, generally using flat holes, to obtain geological data. Should mineralized areas be found, or should favorable structural conditions be indicated, the drilling is followed by drifts and crosscuts. The footage of diamond drilling devoted to this class of prospecting amounts to about 5,000 feet annually.

Within the "main ore zone" the geology and ore areas are projected downward to each new level with sufficient accuracy to permit planning the haulage drift and the "contact" drift. The latter is so named because it is driven on or near the sulphide-schist contact. On the most recent "lift" of new levels the "contact" drifts were driven on the schist side of the contact and on definite bearings, as against the old practice of actually following an irregular replacement contact. This drift is guided by means of occasional diamond drilling, as well as by constant geological observations. Upon completion of this drift, the foot and hanging walls are thoroughly drilled, delineating the ore areas accurately so that the foot and hanging-wall drifts may be driven and the method of mining, with its stope and pillar system, may be planned.

Even after stoping operations have commenced, it is frequently necessary to diamond drill from the stopes with flat or up holes to locate the bottom of hanging pendants of ore which do not reach the level. More often than not the pitch and dip of ore shoots are irregular and must be determined in advance of stoping operations in so far as possible, to permit proper chute spacing. Numerous slips occur, which but rarely limit the ore shoots. Therefore, if a stope is bounded on a side by a slip, it is necessary to prospect beyond the slip at regular floor intervals with either short stope drifts or drill holes until it is assured that a false wall does not exist. To meet the requirements of stope exploration

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a light, compact diamond drill was designed and constructed, weighing about 500 pounds and capable of drilling holes up to 250 feet in depth.

Five diamond drills are operated underground, drilling holes varying from "ES" to "N" in size (7/8 to 2-inch cores). The annual diamond-drill footage is 25,000 to 30,000.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUE

Sampling

Five samplers and a head sampler are employed underground. Faces are usually sampled either by chipping with a sample pick or, to a less extent, by cutting channels. The chips are taken in four lines across the face for the width of the sample, beginning at the back, each line being successively lower. The average size of a sample is about 4 pounds. Individual pieces of rock or ore are limited to 1 inch in diameter. A sample covers a width of 5 feet, this width being regularly adhered to both in drifts and stopes. Usually the height of the wall or face sampled is 7 feet. The average length of a stope round is 6 feet. In the stopes, samples are taken after each round, so that a sample represents a horizontal area of about 30 square feet or about 20 tons.

Sampling errors vary with the type of ore. In the massive sulphides, the chalcopyrite or other ore minerals are very uniformly distributed and the copper content is gradational from the center of the sulphide ore area or the schist contact to the margin of the stopes, where the ore becomes noncommercial. Consequently, in sulphide, the sampling error is very low—an average of 2 per cent above actual content.

In the schist and porphyry ore areas, the occurrence of the ore minerals is extremely erratic, and, though in large areas of schist and porphyry a gradation of copper content may be noted, more often than not it is very rapid and sometimes abrupt. Oftentimes, too, in vertical extent ore and waste may occur on alternate floors for several floors in succession. Naturally then, the sampling error is high and can be reduced only by taking more samples. Possibly, as the ore minerals stand out in the matrix of porphyry and black schist, the personal element enters into the sampling of this class of ore. This source of error, however, is constantly guarded against. The sampling error has varied from 8 per cent to 20 per cent in this class of ore.

Samples are also taken from the various ore bins on the 1000 level or Hopewell haulage tunnel.

The stope samples are checked against the ore-bin samples, and the latter are checked against the smelter assays. In all cases when a discrepancy exists, the stope samples are higher than the ore-bin samples, which in turn are higher than the smelter samples. For the year 1928 the average error for all classes of ore between the stopes and the smelter was 5 per cent.

More specifically the methods and requirements of, and the duties of a sampler underground and in the office are as follows: On entering a stope or heading, the sampler determines the number of samples necessary and records the location of the advance (measured with a tape from known points) in his notebook; he then marks the sample numbers plainly with heavy white chalk on the working face. The marks are made heavy enough to remain on the walls until the sampler on the night shift returns. The sample tags are then filled in,

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showing the location of the sample with regard to a known point, after which the sacks are distributed, and the samples taken.

If slips, dikes, or schistosity are parallel to the face, the sample is taken across the back of the floor below, covering the same area. When the sampler can not get under the back, any dikes or slips occurring on the face are broken into by plugging to permit sampling of the ore. Where it is impossible to obtain a satisfactory sample by the above methods, a series of drill holes, 5 to 10 feet deep, is drilled in the face and the sludges caught and used as the sample.

The samplers are held responsible for ore missed because of areas not being properly marked. The faces or walls must be marked with very heavy chalk either "O. K." or "N. G." The marks are heavy enough to last several days under ordinary circumstances; if the ore is not mined in that time, the faces are remarked.

The sampler is required to make trips to the shaft with samples at regular intervals throughout the shift. This permits the assay office to make determinations and to prepare the daily assay sheet before the night shift goes underground. Sample sheets with the daily stope sample returns are supplied to the foremen and shift bosses of both the day and night shifts. This enables the night-shift bosses to mine according to the results of sampling on the preceeding day shift and prevents the mining of much low-grade or questionable material.

Cooperation with the mining department is essential, and is secured very successfully. The samplers keep in close touch with the foremen and shift bosses whose ideas regarding the value of doubtful ore are very acceptable, and are acted upon immediately by the samplers; the sets or areas in question are carefully checked and rechecked, if necessary. Close contact with the geologists and their stope files enables the samplers to anticipate the general characteristics, peculiarities, and irregularities of the ore bodies.

The samplers keep two sets of assay maps, one in the general office for use by the superintendent and geologist, and the other in the foremen's office for the use of the division foremen and shift bosses. These books are carefully kept and show projected ore outlines, pillar and fence lines, chutes and raises, and the geology. They are posted every day in order to be up to date at all times. Thus the progress of any stope may be followed by watching these maps.

Development work is similarly plotted and posted after every round.

Some stopes are mined by means of vertical holes, and large areas are shot down at one time. In such cases, back samples are taken ahead of the cut, and mining is governed accordingly.

Occasionally samples of broken ore in the stopes, car samples, or chute samples are necessary. It has been determined that for such sampling the proper size to choose, as far as possible, is a piece 3 1/2 inches in all dimensions. Several cars of ore of approximately 1 ton each were chosen from various localities and sorted by hand into different sizes: Minus 1 inch; plus 1 minus 3 inches; plus 3 minus 5 inches; plus 5 minus 7 inches; plus 7 minus 10 inches. Each size was weighed and assayed, and the average assay for each car determined. From the assays it was found that the plus 3 minus 5 inch size was nearest the average for the whole car, and the plus 1 minus 3 inch size second best. From this it was deduced that the proper size to choose must be slightly over 3 inches in diameter.

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Estimation of Tonnages and Values

All underground ores are divided into four classes, according to the gangue: (1) Massive sulphide (sulphide ores with less than 25 per cent silica); (2) silicious massive sulphide (sulphide ores with more than 25 per cent silica); (3) black schist; and (4) quartz porphyry. The proportions of these various types of ore vary in different sections of the ore gody and from level to level. Over the whole ore body the proportions are approximately as follows:

	Per cent
Massive sulphide	44
Silicious massive sulphide	34
Black schist	11
Quartz porphyry	11

The classification "ore blocked out" is used only under the following conditions:

1. Ore between levels when silled both above and below.

2. Ore silled on either top or bottom, with a raise finished and with drifts or crosscuts on top or bottom.

3. Ore with drifts and crosscuts on top and bottom, and with a raise finished, and with more than one diamond drill hole intermediate between levels.

4. In general, all ore where there is practically no risk of failure of continuity.

The classification "probable ore" includes:

1. Ore cut by drifts or crosscuts, but without intermediate work.

2. Ore silled on one level, but without workings of any sort on the level above or below, distance up and down not to exceed width of stope.

3. Ore cut by drifts or drill holes when within the ore zone may be taken up and down a distance equal to the width of the ore.

4. In general ore where there is a risk, yet a warranted justification for assuming continuity.

"Possible ore" includes ore which can not be included in either of the above classifications, nor definitely known or stated in terms of tonnage. This includes ore cut by diamond drill holes outside the main ore zone where good information is lacking.

Tonnage factors in terms of tons per cubic foot are computed from specific gravities. Samples of each class of ore from each stope or ore area are combined into composite samples and the specific gravities of these composites are used in calculating the tonnage factors for the respective classes of ore. In some cases composites from more than one floor are made; the averages of the specific gravities are then used in determining the tonnage factors. If no composites are available, the average for the mine is used. Representative tonnage factors are:

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	Tons per cubic foot	Cubic feet per ton
Massive sulphide	0.1264	7.911
Silicious massive sulphide	.1030	9.709
Black schist	.1101	9.083
Quartz porphyry	.0967	10.341

Three general cases are considered in determining volume. Where the ore continues from level to level, the formula for a truncated cone or prism is used; that is, Area of top + area of base/2 x H = Volume. (H = vertical height). If the ore pinches out to a line, the wedge formula is used; that is, A/2 x H = Volume. Where the ore pinches out to a point, the cone or pyramid formula is used; A/3 x H = Volume. Wherever it is known that the ore pinches or swells between levels, the volume of the ore body is figured in two or more blocks, being split where the dip of the ore changes. In general, the volumes of definitely outlined but irregularly shaped blocks, such as those that contain pillars, are calculated as special cases, adhering to geometric principles.

A correction for dike dilution of both grade and specific gravity is used for all dikes over 1 foot wide. In fire areas, where the ore limits can not be sampled or accurately determined, but are surrounded by old stopes, the wall assays of which show ore, 50 per cent of the area is assumed to have the average value shown by the old assays and 50 per cent of the area to average 3 per cent copper. This, in a rough way, includes those samples approaching our mining limit of 3 per cent copper.

In the calculation of grade, all stope samples are used, making adjustment by the use of wall assays according to the method or direction of mining. Erratic high assays are reduced to the general average of adjacent samples unless local conditions indicate otherwise.

In estimating ore reserves, a detailed knowledge of the principal ore deposit of the mine and the relationship of the smaller ore bodies is requisite. The method of projecting ore outlines is empirically chosen in each case, and therefore can not be stated.

DEVELOPMENT AND MINING

Development

All exploration, development work, stoping methods, tramming, and hoisting equipment are planned with the definite purpose of providing and maintaining an underground production of 3,000 tons of ore per day. To do this it is necessary to develop four new levels every five years, drive 20,000 feet of drifts and raises, and do 25,000 feet of diamond drilling yearly.

The ore body at the United Verde has a definite rake to the northeast, so that in developing new levels in depth the general location of the ore is known. More detailed data prior to laying out the haulage drifts to the ore body is secured in one of two ways - namely, by diamond drill holes drilled down from the lowest level, or by a winze sunk on the sulphide-schist contact, permitting some development during the period of extending and concreting the main shaft. Knowing the general position of the ore body, the main drifts from the shafts are laid out to intersect the center of the ore body, which is then developed as

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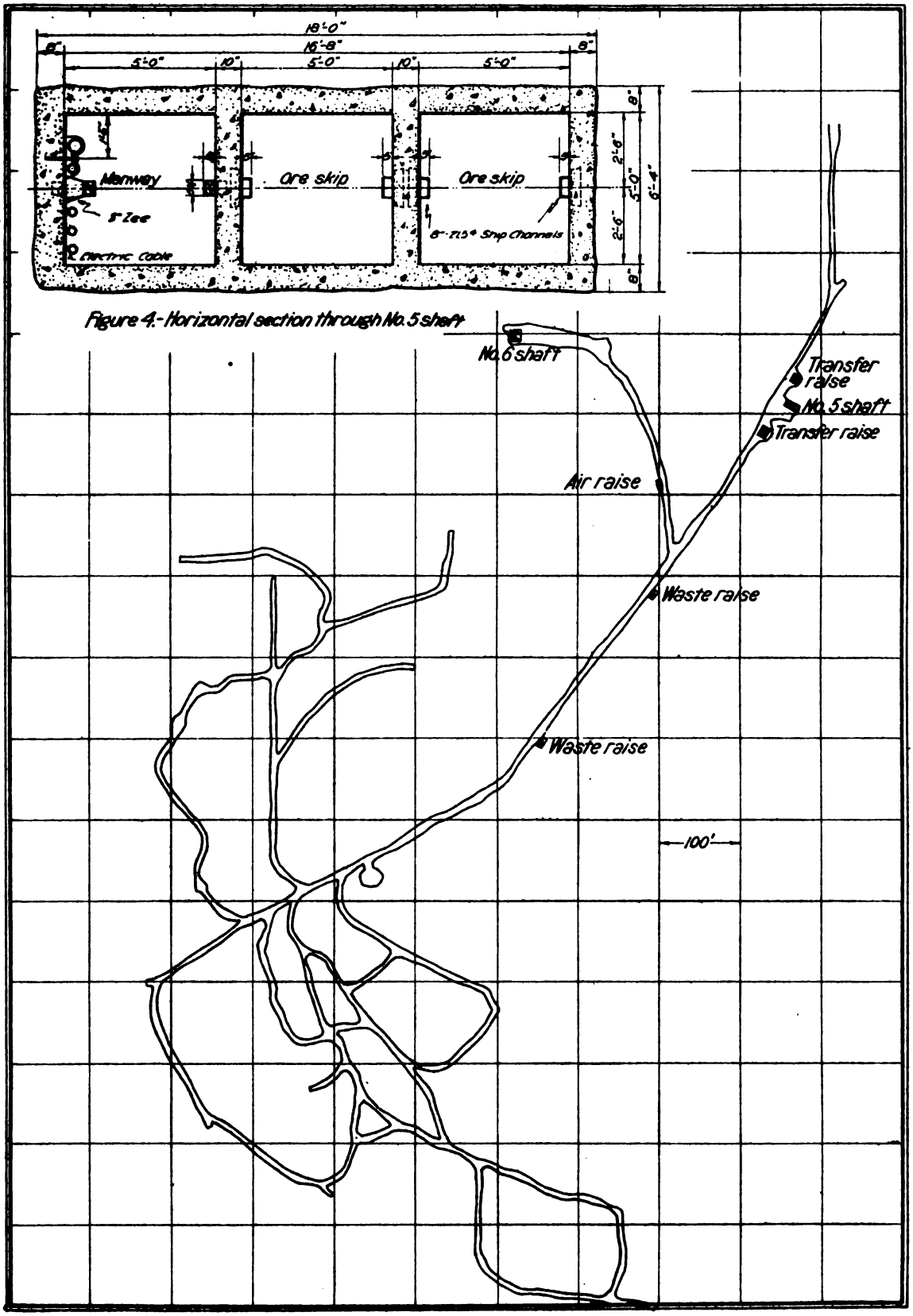

Figure 4.- Horizontal section through No. 5 shaft

Figure 3.- Plan of a typical level

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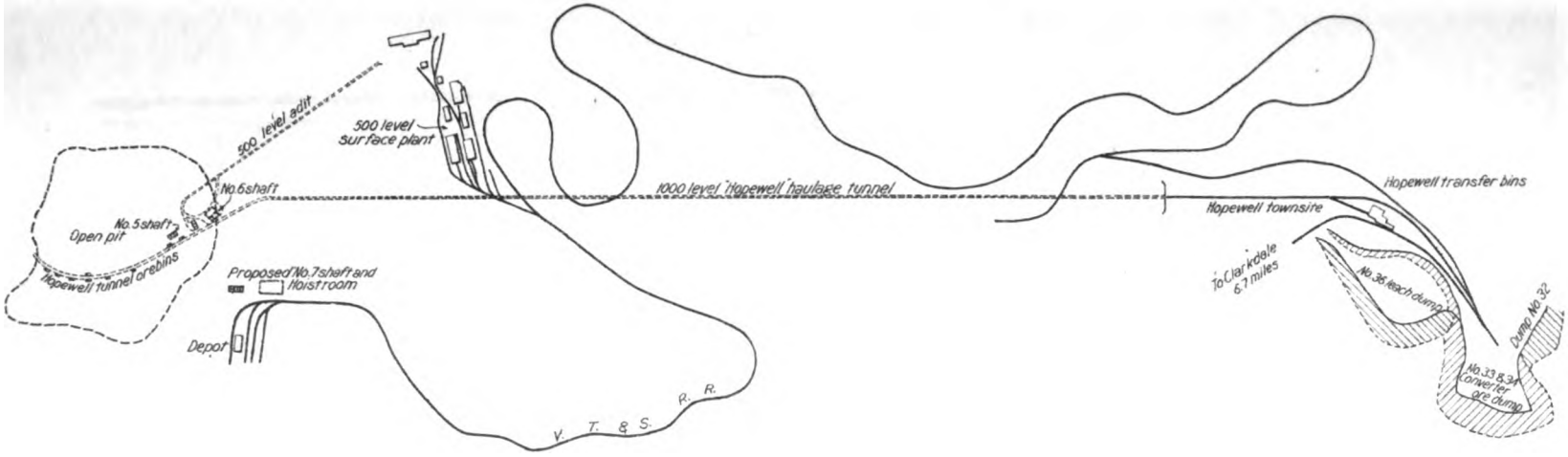

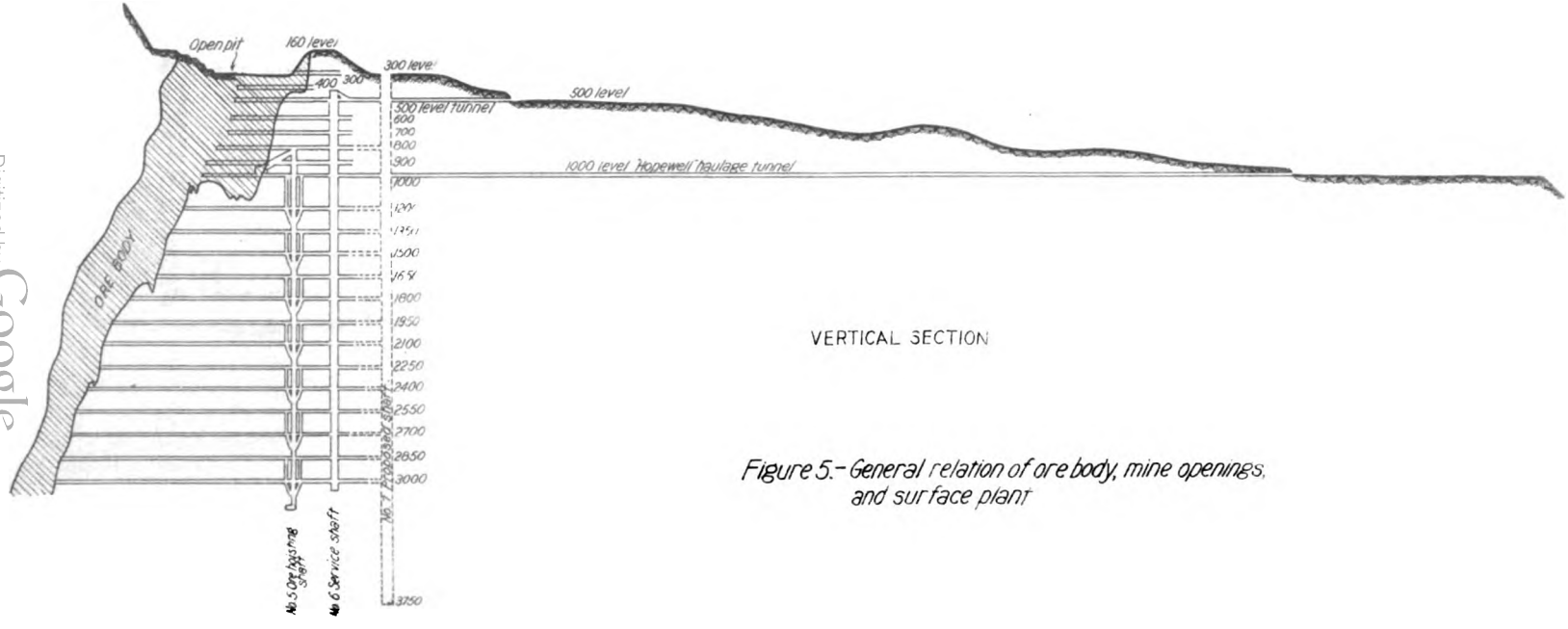

Figure 5.- General relation of ore body, mine openings, and surface plant

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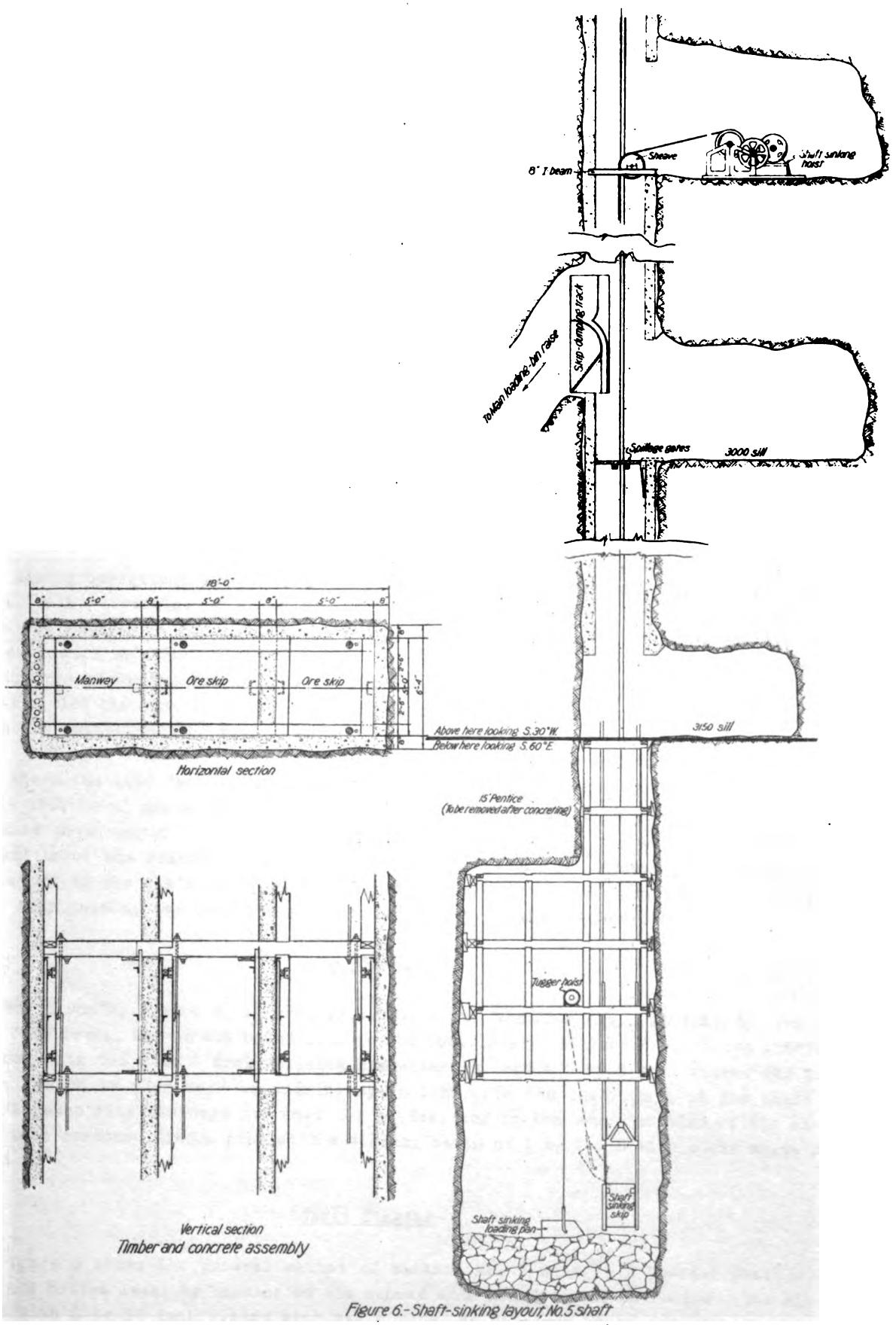

Figure 6.- Shaft-sinking layout, No. 5 shaft

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described under stoping methods. Figure 3, a plan of a typical level, shows the location of the two main shafts with respect to the ore body. Figure 5 shows the general relation of ore body, surface plant, and mine openings.

The mine is served by two main shafts, both of which have underground hoist rooms and are of reinforced concrete construction. No. 6 shaft is used exclusively for supplies and men. It extends from the 500 to the 3000 level, which is the bottom level at the present time, and is connected to the mine surface plant by a 1,600-foot adit. The cage has two decks, 6 feet 9 inches wide by 12 feet 7 inches long, and is double-tracked, permitting timber and steel to be delivered to the various levels with no transfer of materials. The capacity is 100 men, or eight standard trucks.

No. 5 shaft is the main hoisting shaft. All ore from above the 1000 level is dropped by gravity through ore passes to loading bins on that level. Ore from below the 1000 level is hoisted and delivered to similar bins on the same haulage level. Ore from the open-pit operations is also dropped to this level through raises. From here all ore is transported through the Hopewell tunnel by standard-gage, 40-ton cars for a distance of 1.8 miles to outside bins. From there the mill and direct-smelting ore is shipped to Clarkdale over the Verde Tunnel and Smelter Railroad. Low-grade silicious ore, leaching ore, and excess waste from the shovel operations are delivered to dumps by 25 cubic yard dump cars.

Mining operations extend from the 300 level, at an elevation of 5,207 feet, to the 3000 level, at an elevation of 2,528 feet. The sulphide mass above the 900 level has been bulkheaded off because of fire, which started in 1894. The greater portion of the ore above the 600 level will be mined by shovel methods. Border ore outside of the pit limits and fire-country ore above the 900 level will necessarily be mined by underground methods after the pit has cooled the country sufficiently to permit mining operations. For this reason all levels must be maintained and kept open.

Above the 1000 level, the level interval is 100 feet. One 200-foot lift from the 1000 to the 1200 level proved this interval excessive because of chute repairs and increased cost of raise development. Below this horizon the level interval is 150 feet. Formerly, every second level was planned as a trolley motor-haulage level. Recent practice has been to haul the ore to the shaft on every level with storage-battery locomotives. This is more economical than passing the ore through transfer raises to the level below.

No. 5 Shaft

No. 5 shaft, Figure 4, is used exclusively for handling ore. It extends from the 800 to the 3000 level, and is now being sunk to the 3600 level. It is of reinforced concrete construction, with two 5 by 5 foot hoisting compartments, and a 5 by 5 foot manway and pipe compartment, which is also used in sinking operations. In the lower part of the shaft 8-inch, 21-1/2 pound ship channels are used for guides, and in the upper portion of the shaft 3-5/8 by 7 inch seasoned Oregon pine with a wearing strip of 1 by 3-5/8 inch clear white oak is employed.

Shaft Sinking

Figure 6 shows the general method of sinking No. 5 shaft. A 15-foot pentice is left below the bottom level by sinking on the manway side and widening out below. The shaft is timbered with 8 by 10 inch Oregon pine shaft sets, so designed as to permit concreting,

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using 2-inch plank panels for forms. A 32-hole round is used, and the cut placed to take advantage of the ground. The rounds are fired by No. 1 to 10 electric delay detonators using 440-volt alternating current. The crew consists of three shaftmen, one leader, one top man, and one hoistman. The average standard on which a bonus is paid varies from 0.17 feet per miner shift at the beginning of a lift to 0.10 feet at a depth of 600 feet below the bottom level. The following table shows the cost of explosives and labor:

Sinking data, No. 5 shaft

Drilling speed, inches per minute	10
Number of holes	32 to 35
Advance per round, feet	4.5
Powder used, sticks	260
Powder cost per foot advance	$5.20
Labor cost per foot advance	$31.85

The explosive cost does not include fuse or detonators. The labor cost includes timbering and mucking. The water encountered is not serious and averages around 1,000 gallons per hour.

Former practice has been to use a 30 cubic foot sinking bucket, with the usual auxilliary equipment. Present practice is to shovel into an 18 cubic foot pan, which is then raised by means of a small air hoist and dumped into a 38 cubic foot skip. The skip has extended guide shoes to permit lowering it to the bottom of the shaft. The skip dumps automatically into a "dog-hole" leading down to the main skip-loading pocket. The pan is much easier to shovel into than a bucket, and the skip eliminates a second handling on the level above. In addition, broken rock from cutting loading-pockets can be drawn into the skips by gravity.

The work of concreting the shaft is started from the bottom, using 2-inch plank panels for forms. The central mixing station is located on the 500 level, and the concrete is delivered through a 5-inch pipe line. Six hundred feet of shaft was concreted in five months, using 21 men on three shifts. The total cost of concrete per cubic yard was $22.50.

No. 6 Shaft

No. 6 shaft is used only for service (see fig. 7). It extends from the 400 to the 3000 level. Above the 1950 level it is concreted solid, but below 2-1/2-foot concrete rings are placed at 6-foot centers. The cage compartment is 8 by 13 feet, the manway compartment 4 by 9 feet 4 inches and the counterbalance compartment 4 by 3 feet. A sublevel at the collar of the shaft on the 500 level permits loading and unloading of both decks at one time when handling the shift. The shaft is raised from each level in full section, using a double cribbed manway. Concreting is started at the top and carried down as the broken rock is pulled from below. Sectional steel forms are used, supported on 4 by 10 inch sills by means of adjustable hooks from the last set poured. Eighteen men on three shifts will draw the broken rock, trim the walls, set forms, and pour one ring every three shifts. A comparative cost of the solid concrete and the ring methods of construction for No. 6 shaft is given in the following tabulation.

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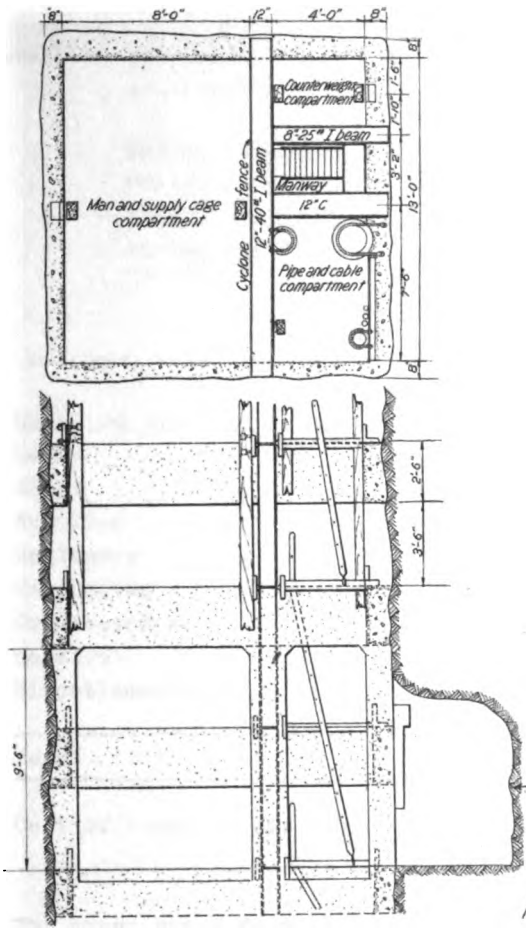

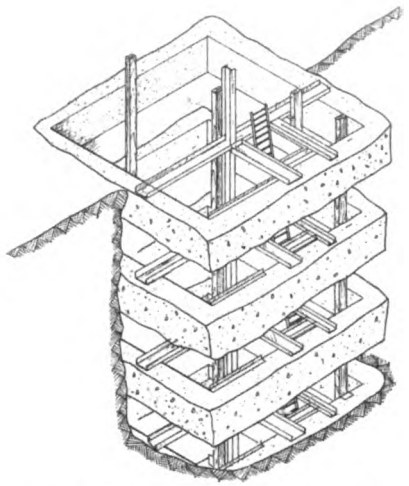

Sectional elevation - showing three rings in place

Figure 7.- No.6 man and service shaft

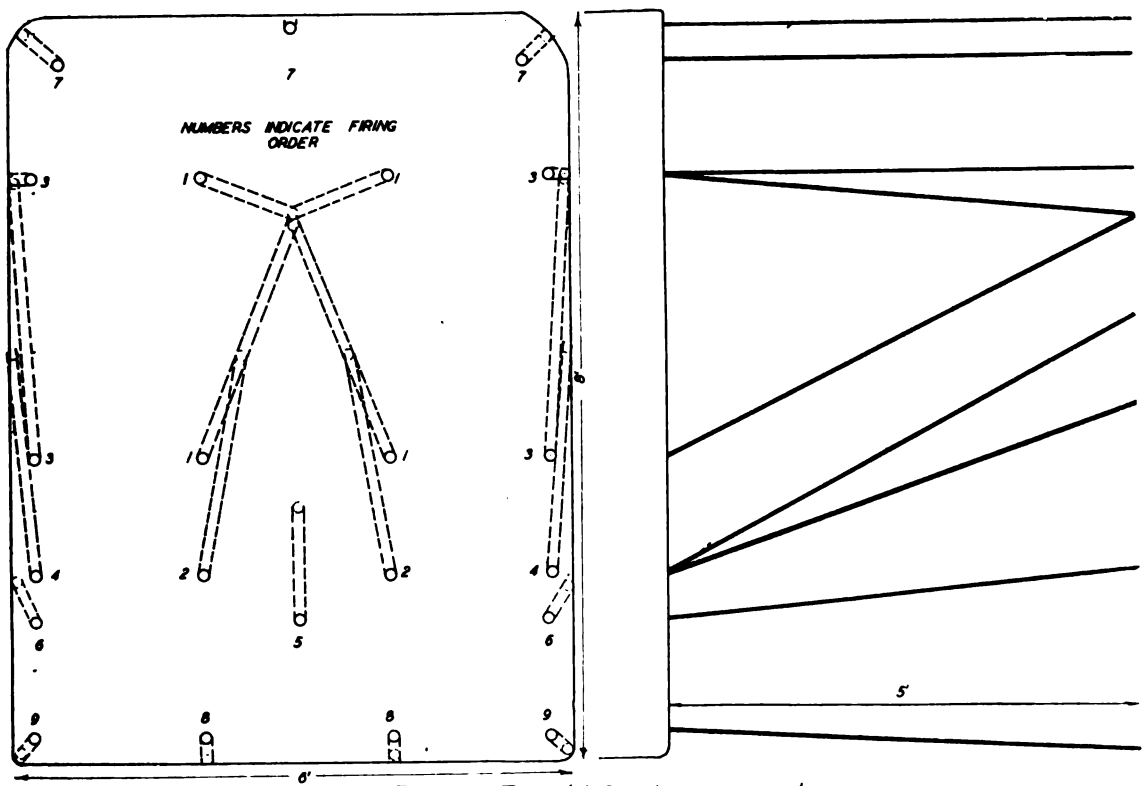

Figure 8.- Typical drift and crosscut round

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Comparison of cost of solid concrete and concrete-ring methods of shaft construction

Method	Concrete mix	Feet	Excavation per foot	Concrete per foot	Total per foot
Solid	1 : 3 : 5	1,550	$68.17	$146.84	$215.01
Ring	1 : 2 : 4	1,050	77.11	50.11	127.22
Saving per foot by ring method					87.79

Detailed costs — concreting by ring method	Excavation per foot	Concreting per foot
Labor	$52.07	$16.60
Shops	0.26	5.90
Supplies	7.68	26.51
Engineers	-	1.05
Explosives	8.47	-
Compressed air	2.08	-
Repairs	6.55	-
Miscellaneous	-	0.05
Total	$77.11	$50.11

Cost of concrete per cubic yard poured. Solid = $61.30; Ring = $58.80.

The above costs do not include steel work, ladders, or pipe lines.

Drilling and Blasting

There is such a wide variation in the class of ground encountered that drilling speeds vary from 4 to 48 inches per minute. The following is a tabulation of drilling speeds in the various formations, taken with a CP-6 drifter:

Rock formation	Drilling speed, inches per minute
Massive sulphide	4 to 8
Silicious massive sulphide	6 to 10
Black schist	24 to 48
Porphyry	10 to 30

The wide variation in drilling speed in the black schist results from the different degrees of alteration of the rock, whereas in the porphyry it is caused by the varying amounts of quartz.

The breaking qualities of all the rocks depend on many local geological factors and vary greatly. No definite statement can be made concerning any of them except that the black schist usually breaks quite fine. The breaking quality of the massive sulphide is apparently affected most by the copper content: The higher the copper content the finer the fragmentation. The siliceous massive sulphide seems to follow no rules. The porphyry, al-

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though not the hardest rock, is undoubtedly the toughest and the most difficult one in which to get a good fragmentation except where it is altered or already broken by small slips or joints.

A 145-pound mounted drifter is used for practically all drifting and stoping. Several makes of stopers are in use for raising, both hand and self rotated. Two makes of light (38 and 52 pound) jackhammers are used for plugging and a 65-pound hand-held machine for sinking. All drills are of the wet type.

Three sizes of drill steel are used: 7/8-inch quarter-octagon for jackhammers and stopers in soft ground; 1-1/4-inch hollow-round for all Leyner-type machines; and 1-inch quarter-octagon for stopers in hard ground. The double-taper cross bit with 5 and 14° taper is standard for all steel. Starters are 30 inches long, with 1-7/8-inch gage. Length changes are in multiples of 10 inches, with 1/16-inch gage change. Columns and crossbars are made from 3-inch extra heavy pipe.

Two sizes of hose are used: 1/2 inch for water, and 1 inch for all air hoses. All hose is provided with standard lugged couplings and spuds which fit all types of machines and 1-inch pipe lines. Damaged hose is brought to the surface for repairs and mended by means of a compressed-air device, which cuts the hose, inserts nipples, and wraps the joint with wire. Another mechanical device forms the wire loops for wrapping joints.

Fifty per cent strength gelatin dynamite in 1-1/8 by 8 inch sticks is used in sulphide and porphyry, and 35 per cent in schist and for all plugging. A smooth, black-finish, cotton-countered safety fuse is used in 4 and 5 foot lengths for blockholing and in 7 to 9 foot lengths for development and stoping. No. 6 and No.8 detonators are used with 35 and 50 per cent strength gelatin dynamites respectively. Electric blasting is employed in all shaft sinking and in drifts starting from shafts and winzes. Primers are delivered to the underground fuse magazines in wood-lined containers made of carbide cans and provided with hinged covers. Stemming consists of a clayey silt in paper cartridges and is used in all hard ground.

Bowlders are blasted during lunch hour, and stope and drift rounds are blasted only when the men are going off shift. An electric signal system between levels serves to eliminate the danger of blasting by a clearance signal from the level above. This signal system is essential because of the danger from sulphur dioxide and hydrogen sulphide formed by blasting in heavy sulphides.

Drifts and Crosscuts

Main drifts and crosscuts are carried 6 by 8 feet in cross section in untimbered sections and 5 by 7 feet where timbered. Gangways under stopes, when timbered, are carried 5 by 10 feet in the clear. When air conditions are conducive to decay, the timber is treated with zinc chloride. Only about 5 per cent of the drifts require timbering at the time of driving, and these are drifts in the area of the Verde fault and a few drifts in greenstone. Drift sets are usually made of 10 by 10 inch Oregon fir, but occasionally 8 by 8 inch or 12 by 12 inch is used, as required by the pressure.

The number of holes and type of round vary greatly, depending on the character and hardness of the ground. The rounds most commonly used are the V-cut, toe cut, and upper cut. In any of these rounds, the number of holes will vary from 10 to 28. Figure 8 shows a typical round for a 6 by 8 foot drift.

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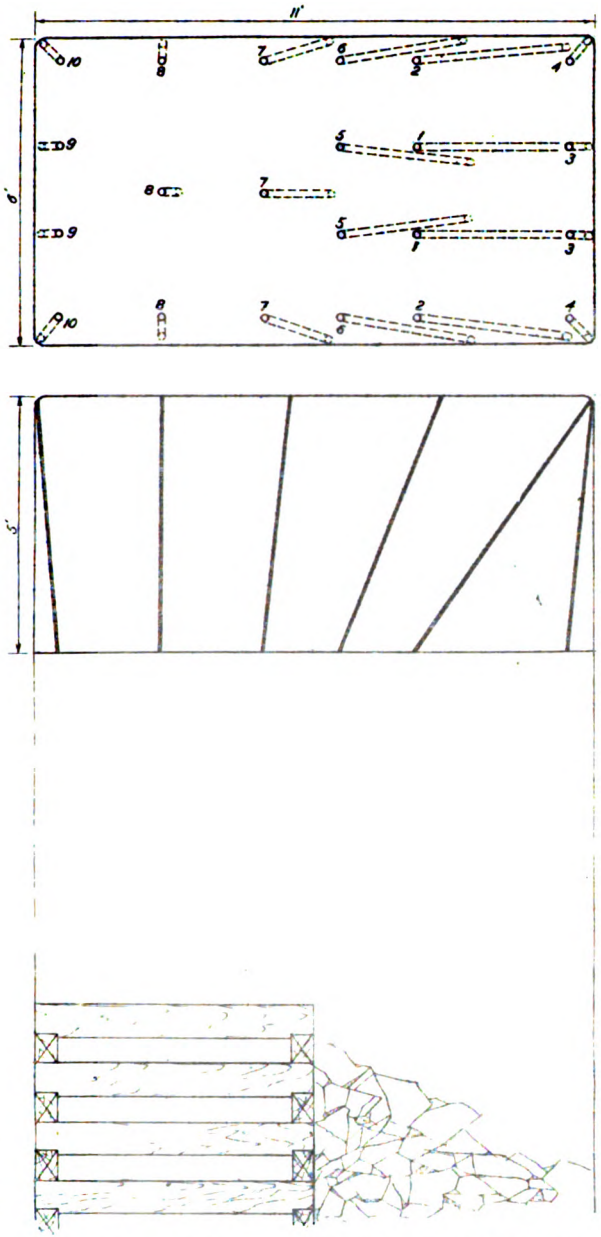

Figure 9.- Raise round. Numbers indicate firing order

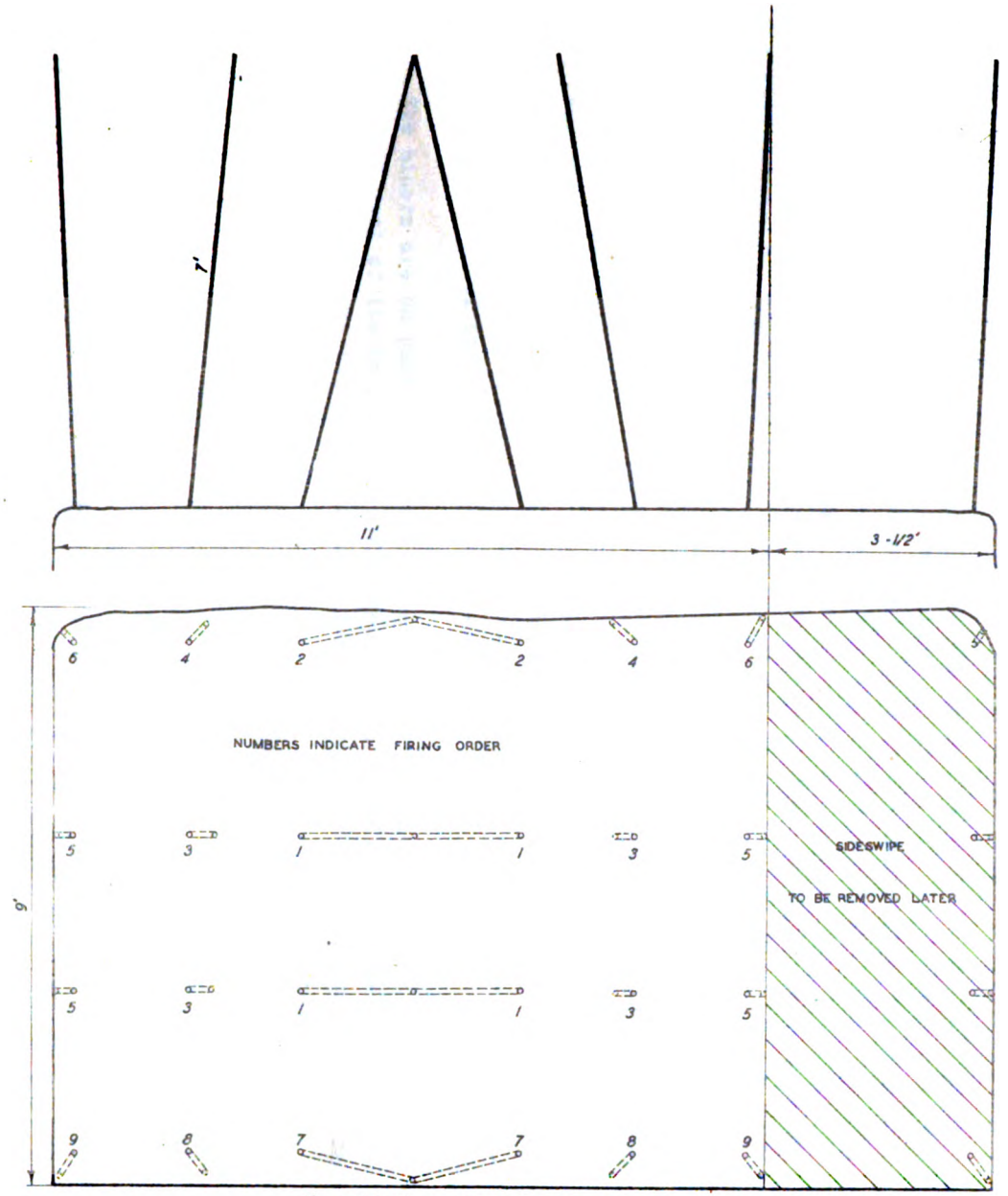

Figure 10.- Hopewell haulage-tunnel round

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All drifts are worked two 8-hour shifts per day. A miner drilling with one machine, using a vertical bar, usually drills and blasts in one shift. Two muckers clean the drift on the opposite shift. In extremely hard sulphide it occasionally requires as much as four drilling shifts to get in a round. The average drift requires 136 sticks of 1-1/8 by 8 inch 50 per cent strength gelatin dynamite to the round. Drilling and breaking in drifts are done on contract, which includes labor and explosives. The mucking is done under the bonus system.

Raises

Raises between levels are 6 by 11 feet in section, and the manway side is carried up with single native-pine 5 by 8 inch cribbing with 56-inch inside clearance. The number of holes required depends on the character and hardness of the ground, but will average about 22. The cut is drilled over the chute side, as shown in Figure 9. Material is handled to the face by either a column-type Little Tugger or an Anaconda-type air hoist, used with a 12 by 14 inch by 4-foot steel bucket and 3/8-inch steel cable.

The miners are on contract; they drill and blast two rounds, then raise the cribbing to within 7 feet of the back. The regular motor crews draw the broken material on the opposite shift from the miners. Two miners work in each raise one 8-hour shift daily. The average round requires 144 sticks of 1-1/8 by 8-inch 50 per cent gelatin dynamite.

Adit

The main adit on the 500 level, which connects the surface plant to No. 6 shaft, is 1,600 feet long; 400 feet is untimbered and the remaining 1,200 feet is timbered with 10 by 10 inch Oregon pine sets, 8 by 9 feet in the clear, spaced on 5-1/2-foot centers. All timber and lagging have been partly covered with diamond-mesh wire and gunited for fire protection.

Hopewell Tunnel, on the 1000 level, is the main ore-haulage level, and connects the underground and shovel pit with the outside transfer and storage bins on the V. T. & S. Railroad. The adit is 10 by 13 feet in the untimbered portions; 2,700 feet is timbered with 10 by 10 inch Oregon pine, at 5-1/2 feet on centers. The sets are 9 feet high, with 10-foot caps; the posts are battered to give a 12-foot width at the sill.

The adit is now being extended and the heading is carried 10 by 11 feet and is untimbered. The width is limited by the amount of broken rock which may be gotten out in two shifts. The additional 3-1/2 feet will be "sideswiped." Figure 10 shows the round used. The 10 by 11 foot heading is drilled with two machines mounted on separate columns, the drilling and blasting being done on one shift. The round is cleaned out on the opposite shift with a mechanical loader. Approximately 136 sticks of 1-1/8 by 8 inch 50 per cent strength gelatin dynamite are used to the round.

Mechanical Loading

A portable loading dock, equipped with a double drum, column-type hoist, with a hoe-type scraper, has been used for loading in drifts, but has not proved entirely satisfactory. It is found more economical to load by hand in drifts which can be cleaned in one shift by two men. A compressed air operated mechanical loader is used in large drifts, pump sumps, hoist rooms, and occasionally in 6 by 6 foot headings.

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Scrapers have been used to good advantage in stope sills where the chute spacing is irregular, and in spreading waste under favorable conditions. The two-drum air hoist has been replaced by a three-drum electric type, using two cables on the pull back. This permits the scraper to be moved in any position without constantly changing the tail block. Because scrapers under certain conditions tend to tear up the plank flooring, they have not been used to any extent in stoping operations.

Stoping

Although several mining methods are employed in the mine, over 85 per cent of the ore is extracted by two methods, namely—horizontal cut-and-fill and horizontal square-set. The other methods are used for exceptional conditions, such as in the recovery of level and vertical pillars and small, odd-shaped, isolated ore bodies.

The table below gives the percentage of ore extracted by the various methods in 1929:

Horizontal cut-and-fill	61 per cent
Inclined cut-and-fill	3 per cent
Horizontal square-set	25 per cent
Inclined square-set	2 per cent
Shrinkage	1 per cent
Top slice	8 per cent
Total	100 per cent

Horizontal Cut-and-Fill

In the development of a level, regardless of the stoping method to be used, a drift is driven along the sulphide-schist contact. The former practice of following the contact resulted in a very crooked drift, with difficult haulage conditions. As stated before, present practice is to drive in the schist, near the contact and roughly paralleling it by the use of easy curves and short tangents. From this drift the necessary raises and chutes are run for stoping operations. In some places the ore body is so wide that two parallel drifts are necessary to secure the proper chute spacing in the stopes. One of these parallel drifts is necessarily driven in the hard massive sulphide at a high cost per foot.

The practice at one time was to sill out either on the level or 10 feet above the track, and then put in a timbered gangway and chutes. This involved a high first cost for timbering and also a high maintenance cost, and was discontinued in cut-and-fill mining, although a somewhat similar procedure is still followed in square-set mining. The more recent practice is to drive an untimbered drift on the level and sill the stope at from 21 to 25 feet above the rail, depending upon the strength of the ground over the back of the drift (see fig. 11).

Due to the great variation in the width of the ore body and the irregularity of ore outlines, it is not possible to establish any fixed dimensions for the cut-and-fill stopes. They are all opened up to what is considered the maximum size consistent with safe mining conditions, both as regards the workmen and the physical condition of the mine. This varies from 30 to 160 feet across the ore body and from 60 to 200 feet along the strike. These dimensions are also affected by the necessity for maintaining regularity in the vertical pillar system (see fig. 12).

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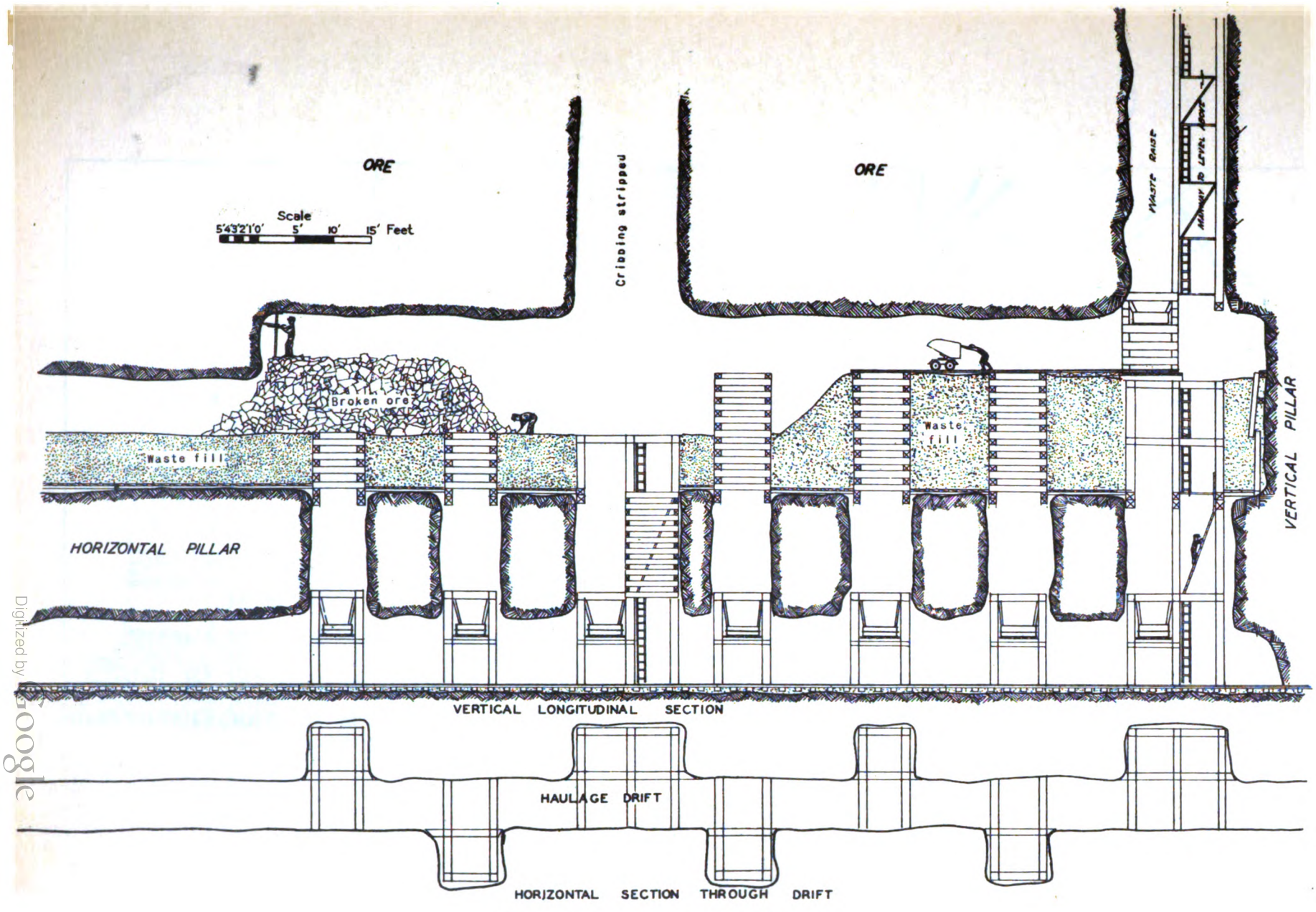

Figure 11.– Typical cut-and-fill stope

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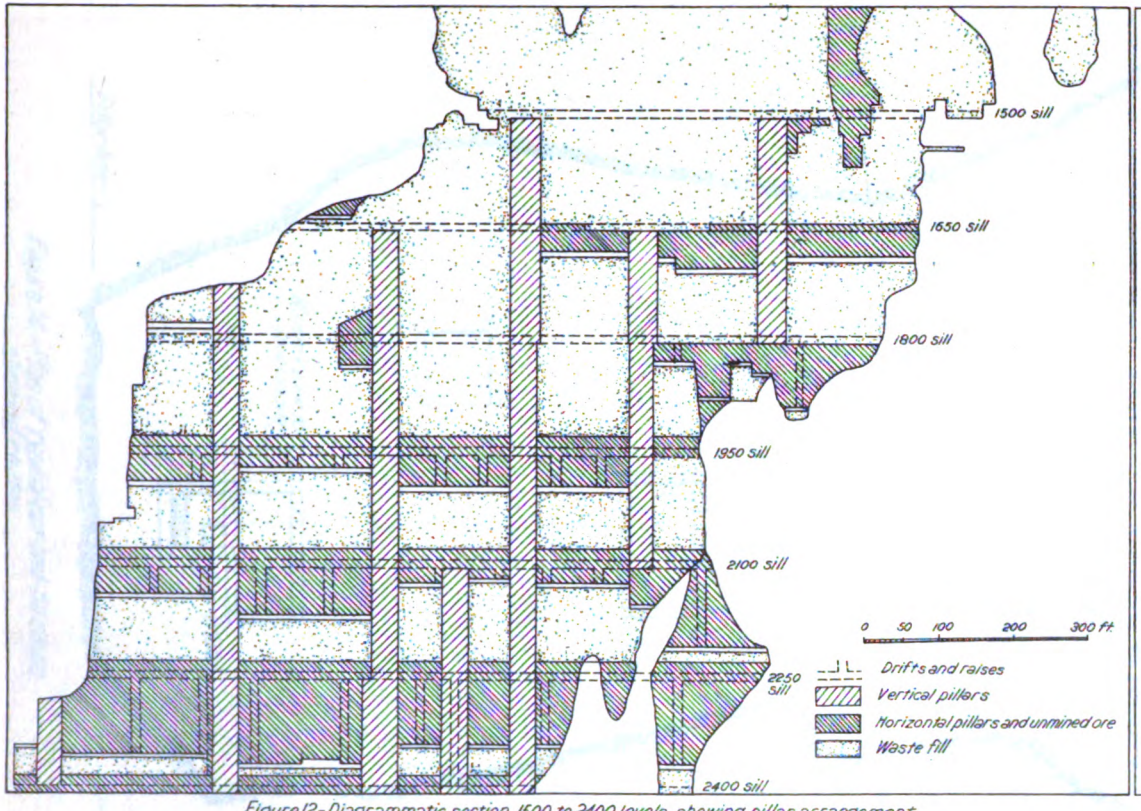

Figure 12.-Diagrammatic section 1500 to 2400 levels, showing pillar arrangement

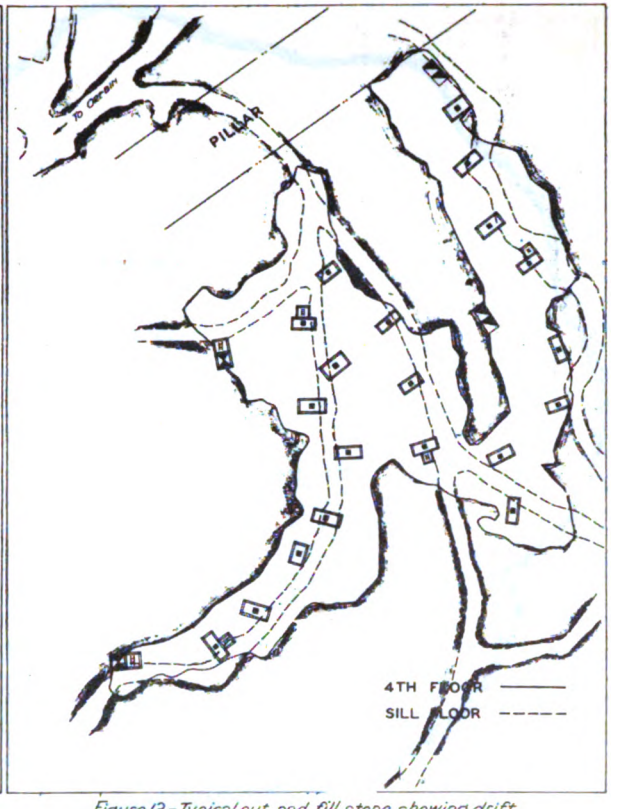

Figure 13.-Typical cut-and-fill stope, showing drift, chute, and raise layout

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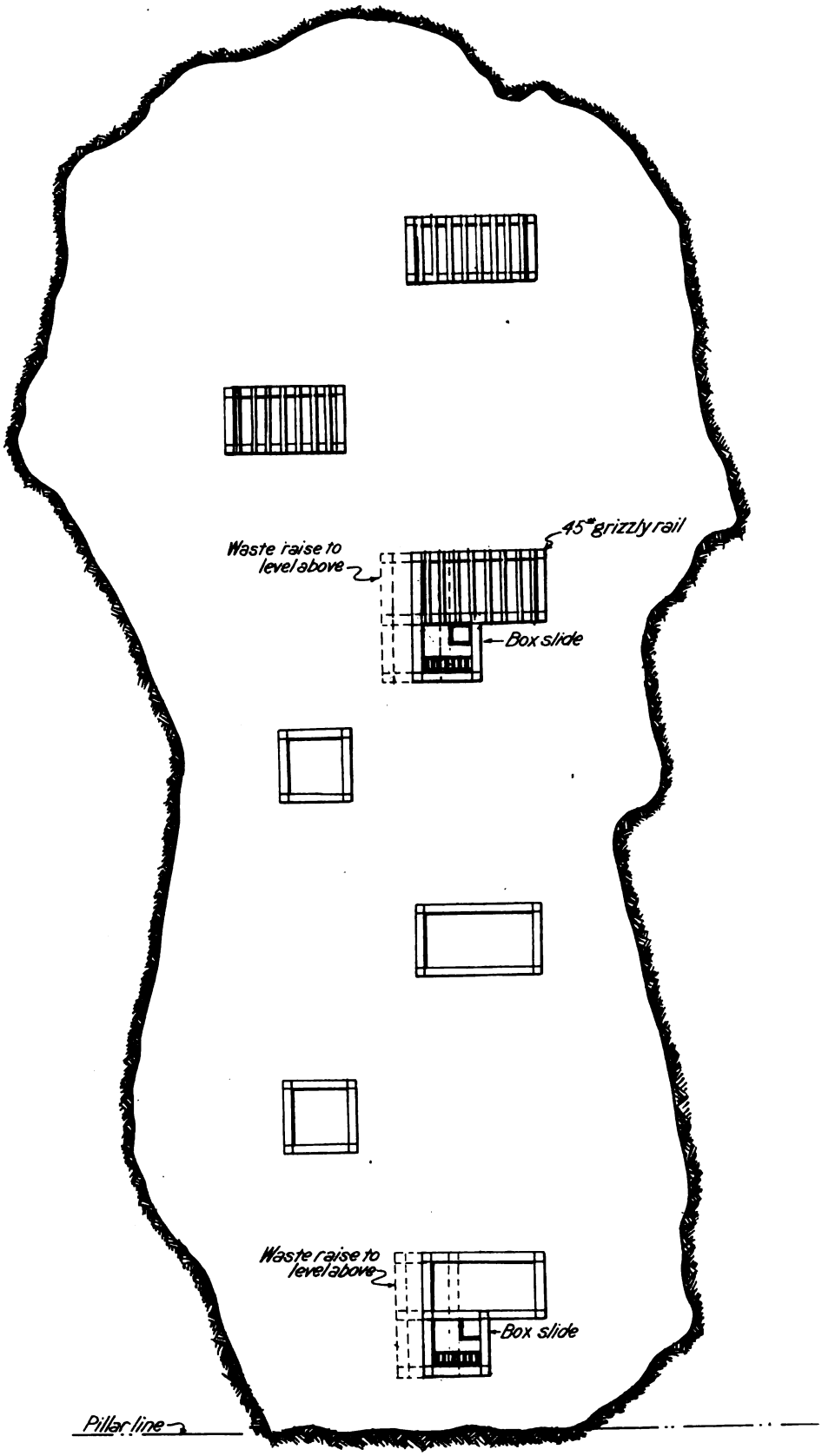

Figure 14.-Plan of typical cut-and-fill stope, showing chutes

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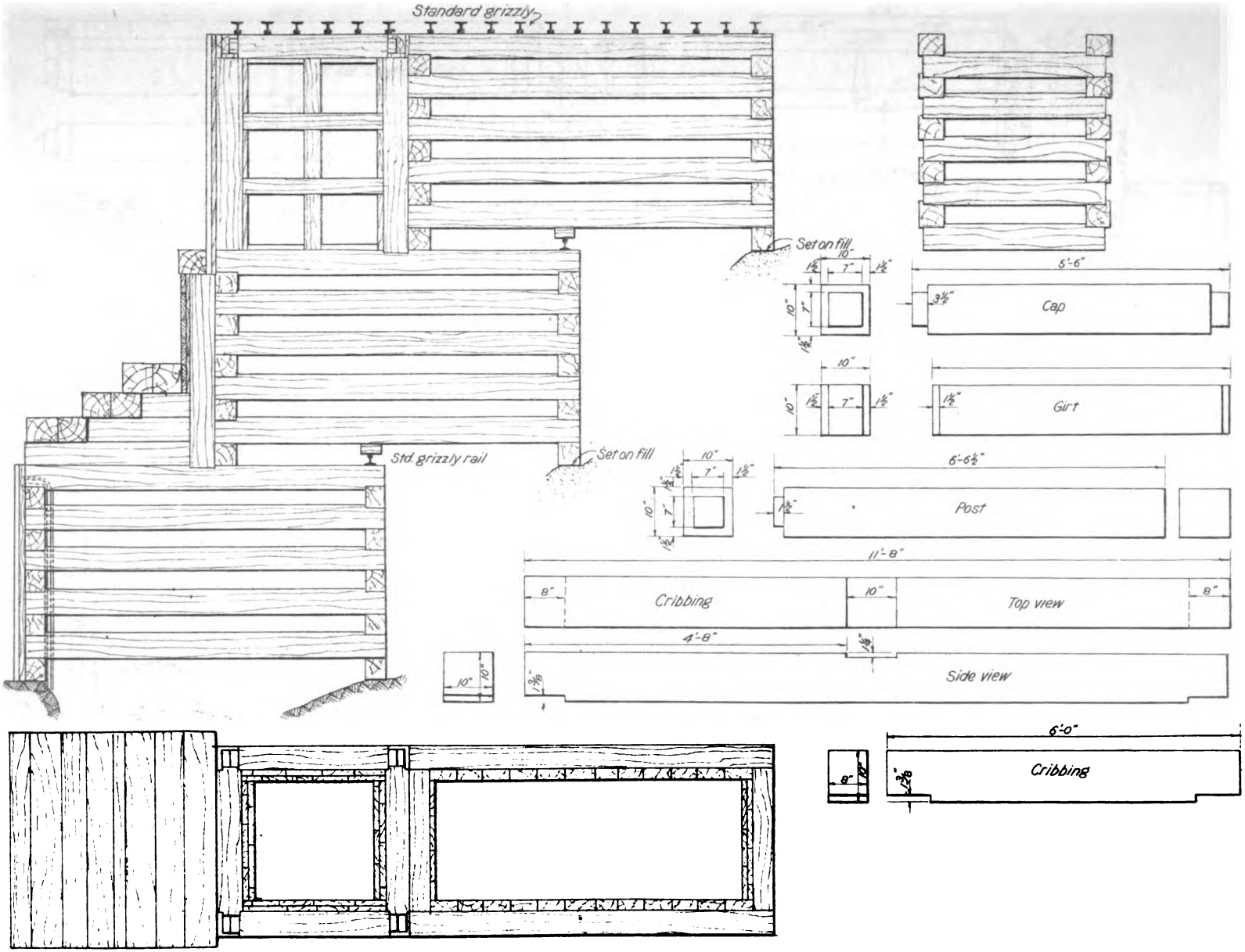

Figure 15 - Standard offset chute

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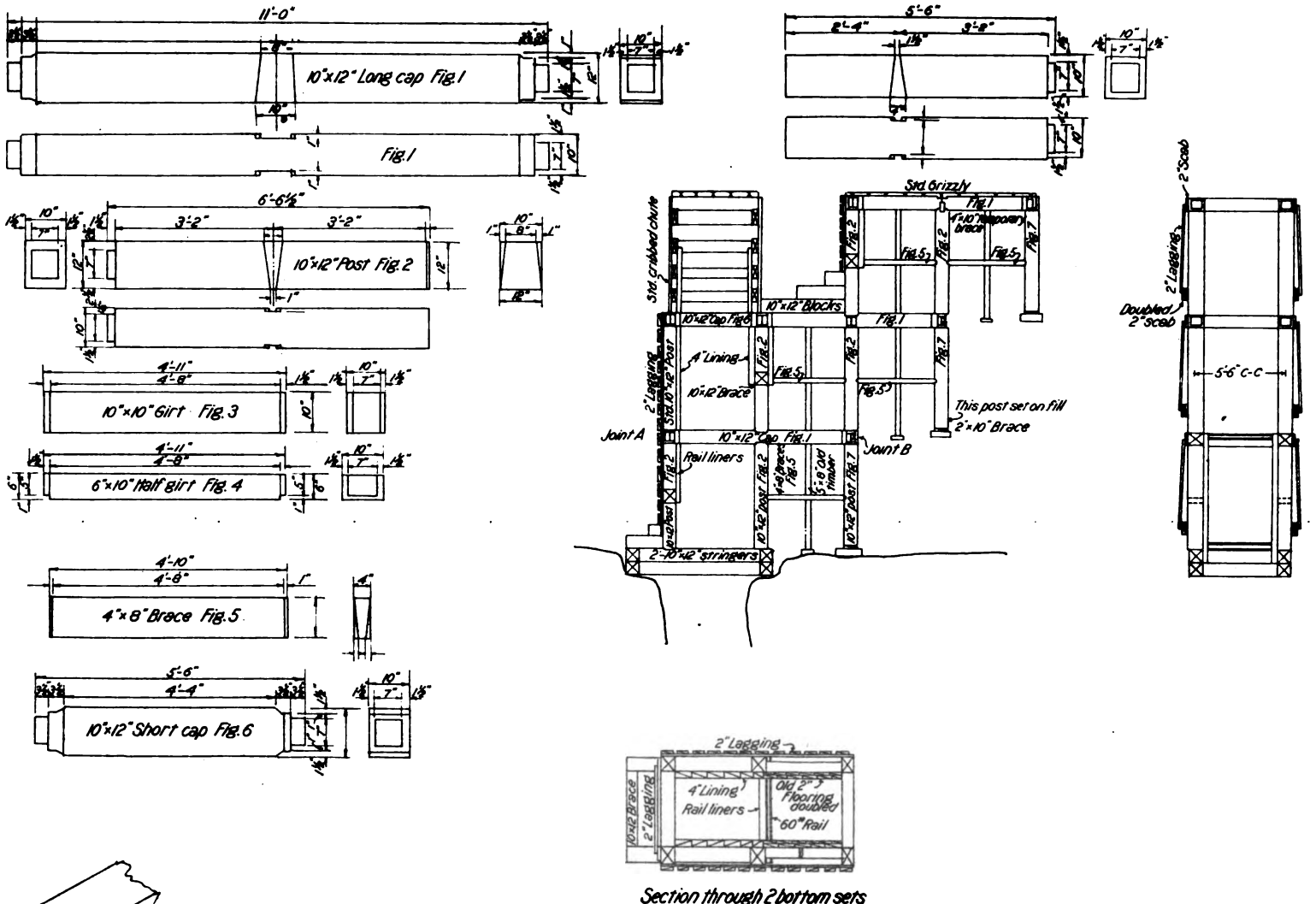

Figure 16.- Six post offset chute using square-set

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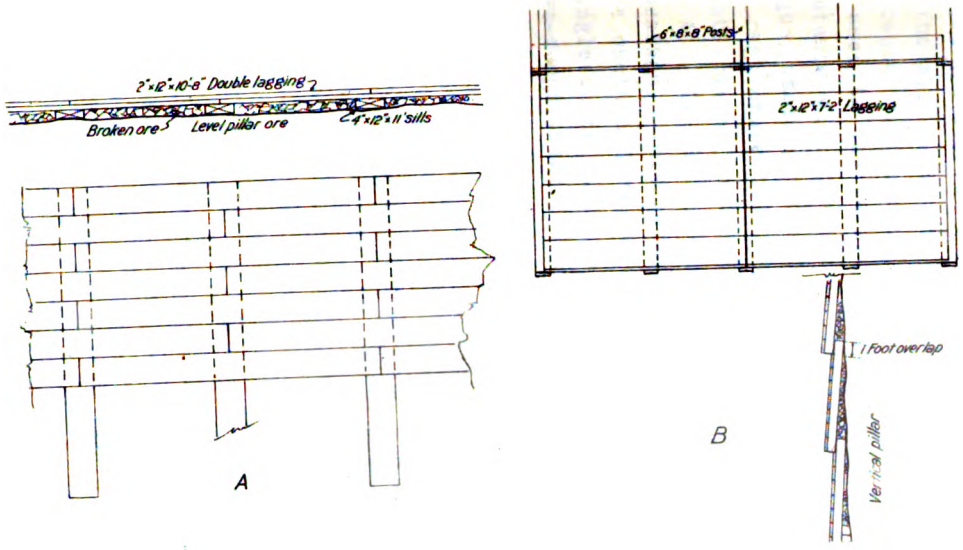
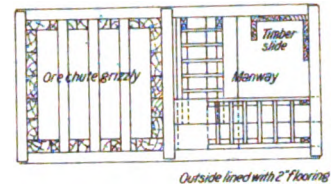
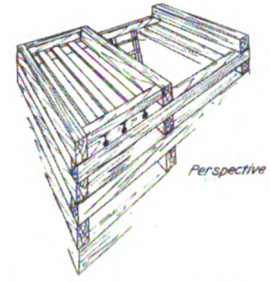


Figure 17.- A, Sill flooring; B, Pillar fence





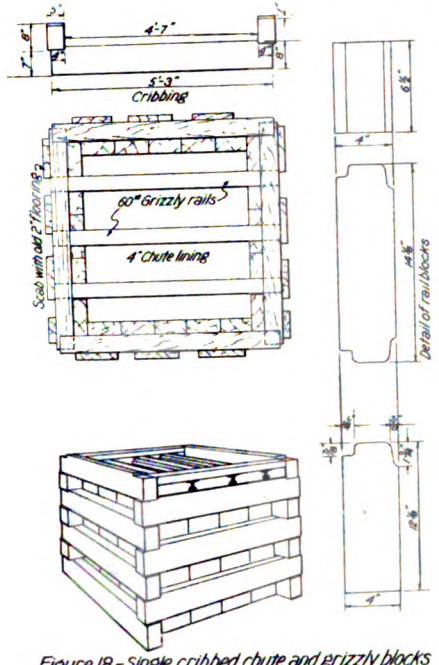


Figure 18.- Single cribbed chute and grizzly blocks

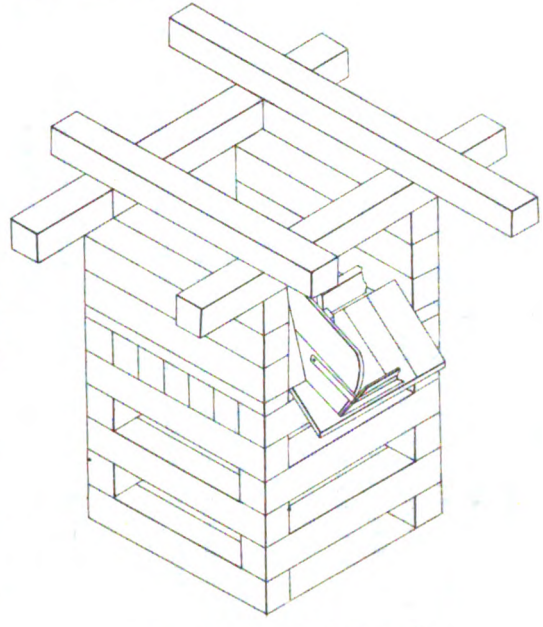


Figure 19.- Standard waste chute

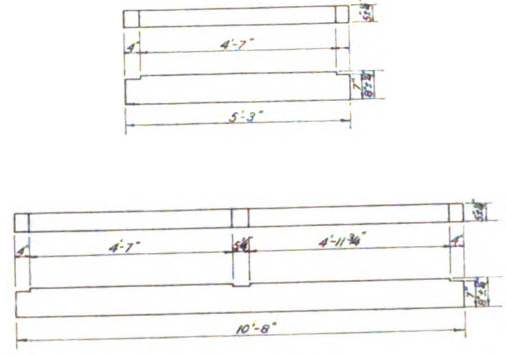


Figure 20.- Detail of framing, standard 2 compartment cribbed chute

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The method employed in laying out and preparing a block of ore for stoping is, first, to drive the contact drift and determine by diamond drilling the approximate ore boundaries. After this is done, the vertical pillar scheme and the size of the stopes is determined. The next step is to drive a raise, which will later serve as a waste raise, through to the next level. This is a standard 6 by 11 foot raise, with a cribbed manway compartment, the chute side being left untimbered. This raise is usually located as near the center of the stope as possible, as shown in Figure 13. In the smaller stopes only one waste raise is used, but in stopes 100 feet or more in length, two raises are used in order to insure continuous production and to provide better ventilation, the second raise being placed near one end of the stope (see fig. 14).

Where two raises are used, the timber is stripped from both of them, but where only one is used, the timber is left in. This insures a ventilation outlet from the stope through the manway compartment even while the raise is being used for filling. In the case of the two-raise stope, only one of the raises at a time is used for waste fill, the other being left open for ventilation, and they are used alternately to meet the requirements of filling.

After completion of the waste raise, silling operations are started from this raise 21 to 25 feet above the rail. The sill floor is made 7 feet high. At the same time chute raises are started from the drift and driven 1 or 2 feet above the level of the sill, so that as the sill advances the raises are easily found. The chute spacing is 16-1/2 feet center to center in the massive sulphide and 22 feet in the schist and porphyry. The closer spacing is used in the massive sulphide for two reasons: The ore is very heavy and the throwing distance with the shovel must be reduced; also, it breaks into fragments with very sharp edges, causing excessive wear on timbered chutes. Double 10 by 12 inch stringers, upon which to rest chute cribbing or one type of offset chute (figs. 15 and 16) are placed over the chute raises so that the top of the upper stringer is level with the stope flooring (fig. 17 A). On these stringers is placed a grizzly of 60-pound rails, ball down, with 11-inch openings. This grizzly is for use in mining the second cut, and is removed before raising the chute and replaced for each successive cut. Figure 18 shows the method of timbering a single chute, and the spacing blocks for grizzly rails. Figure 20 illustrates the standard two-compartment cribbed chute.

After the sill has reached the pillar lines and the ore limits, a second 7-foot cut is started from the waste raise. In stopes having two waste raises, the cut is started from the raise near the end of the stope. All drilling is done with a drifter, using two rows of flat holes for the 7-foot cut. As many holes as can be used effectively are drilled from each set-up by fanning them out. These holes average about 7 or 8 feet in depth. They are loaded with 50 per cent strength gelatin dynamite, and detonated with No. 8 blasting detonators.

The ore, particularly the massive sulphide, breaks with about 30 per cent of the fragments too large to pass through the 11-inch grizzly opening, thus necessitating considerable secondary blasting. Spacing the holes closer in the primary blasting has been tried with the hope that it would give a better fragmentation, but it resulted only in a lot of cut-off holes and much powder left in the muck pile. The secondary blasting is done with 35 per cent strength gelatin dynamite and consumes 20 per cent of the total explosives used.

For each cut a temporary shoveling floor of 2-inch planks 5 feet 4 inches long is laid. This floor is recovered after shoveling is completed. The broken planks, which do not exceed 10 per cent of the total, are used in slabbing the outside of the chutes, and the unbroken boards are used again in the next floor.

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After two or three rounds have been blasted on the new cut, shoveling operations are started and follow at from 10 to 25 feet behind the mining to the end of the stope. About 2 per cent of the ore broken is sorted out as waste by the shovelers, who are allowed 90 cents a ton for waste sorted. As soon as the shovelers pass a chute and there is no more ore to go into it, the timbermen raise the chute 7 feet and filling is started. To start filling, enough waste fill is dumped down the waste raise directly into the stope, and leveled off by hand, to form a bench large enough for the erection of a bulkhead waste chute (fig. 19). Filling is done by hand-tramming in specially designed scoop cars (fig. 21) run on sectional track (fig. 22) laid on the fill.

Before introducing the first fill, a sill floor is laid, consisting of 4 by 12 inch sills at 5-foot 4-inch centers, and a double floor of 2-inch planks 10 feet 8 inches long placed so as to break joints.

Along the pillar line a gob fence (fig. 17 B) is built on each floor before the introduction of waste. This fence is built of 6 by 8 inch posts 8 feet long, spaced 3 feet 7 inches center to center, covered with 2-inch planks 7 feet 2 inches long with staggered joints.

The vertical section shown in Figure 11 illustrates the sequence of operations in cut-and-fill mining.

Filling

The material used for filling is the waste rock from stripping operations in the shovel pit and the waste from underground development work. It consists mainly of diorite and porphyry, some of it being quite fresh and consequently coarse, with enough fines to fill all voids. This waste is delivered to the various levels through two main waste raises, one directly from the shovel pit and the other from the Hopewell haulage tunnel. Because of the distance the waste falls in these raises, which are intentionally kept empty at the top, considerable crushing action takes place; the waste as delivered to the stopes is considerably finer than when dumped into the tops of the raises. Only waste which is practically free from sulphur is used, a necessary precaution on account of the great fire hazard in this mine.

From the main waste raises the waste is trammed by the same equipment that hauls the ore, to the tops of the waste raises leading from the stopes, and is spread in the stopes as described above.

No effort is made to tamp the waste. The action of the water from the drills seeping downward and the force of the falling ore from blasting cause a settlement of about 8 inches for each 7-foot layer of fill. Subsequent settling in old stopes is very slight. The result is a well compacted waste fill which offers very little difficulty in later mining operations in the removal of either the level or vertical pillars.

Inclined cut-and-fill stoping has been used in a few special cases where the conditions seemed to warrant the change from the horizontal method, as for example, in the extraction of level pillars where the weight on a flat back and square brow had to be relieved or square-setting resorted to, or in a few isolated ore bodies whose general shape makes inclined work preferable to horizontal.

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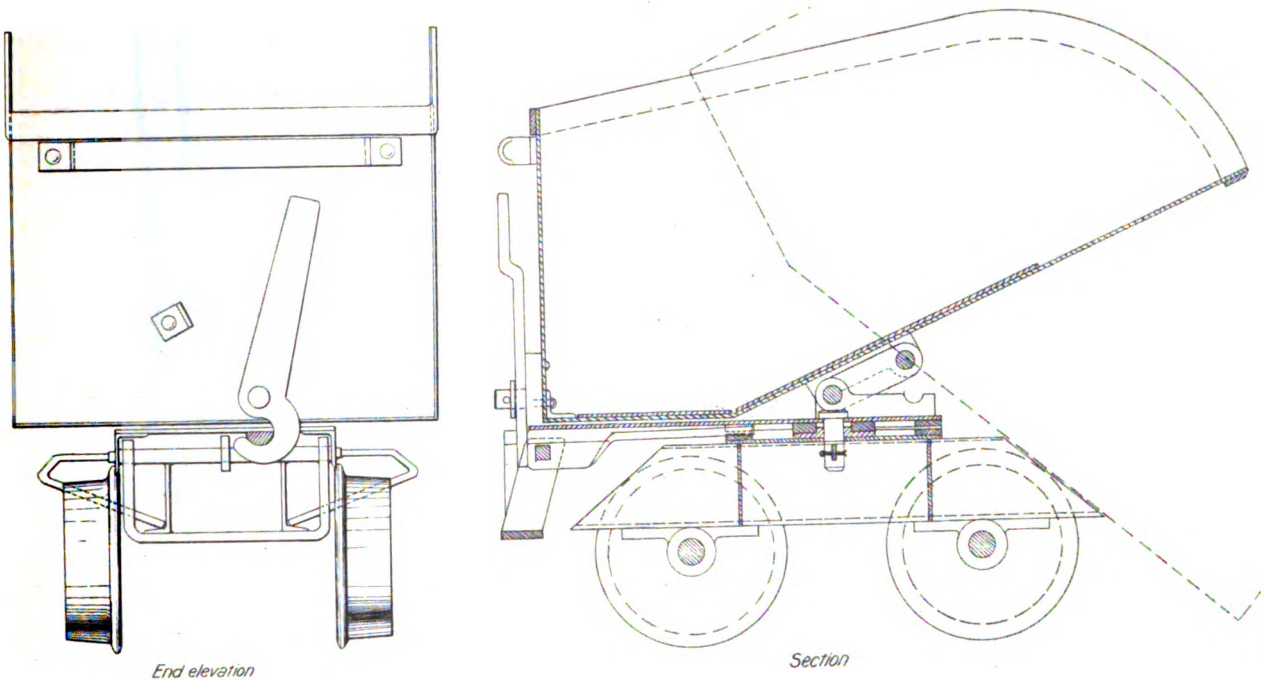

Figure 21.-18 cu.ft. scoop-body car

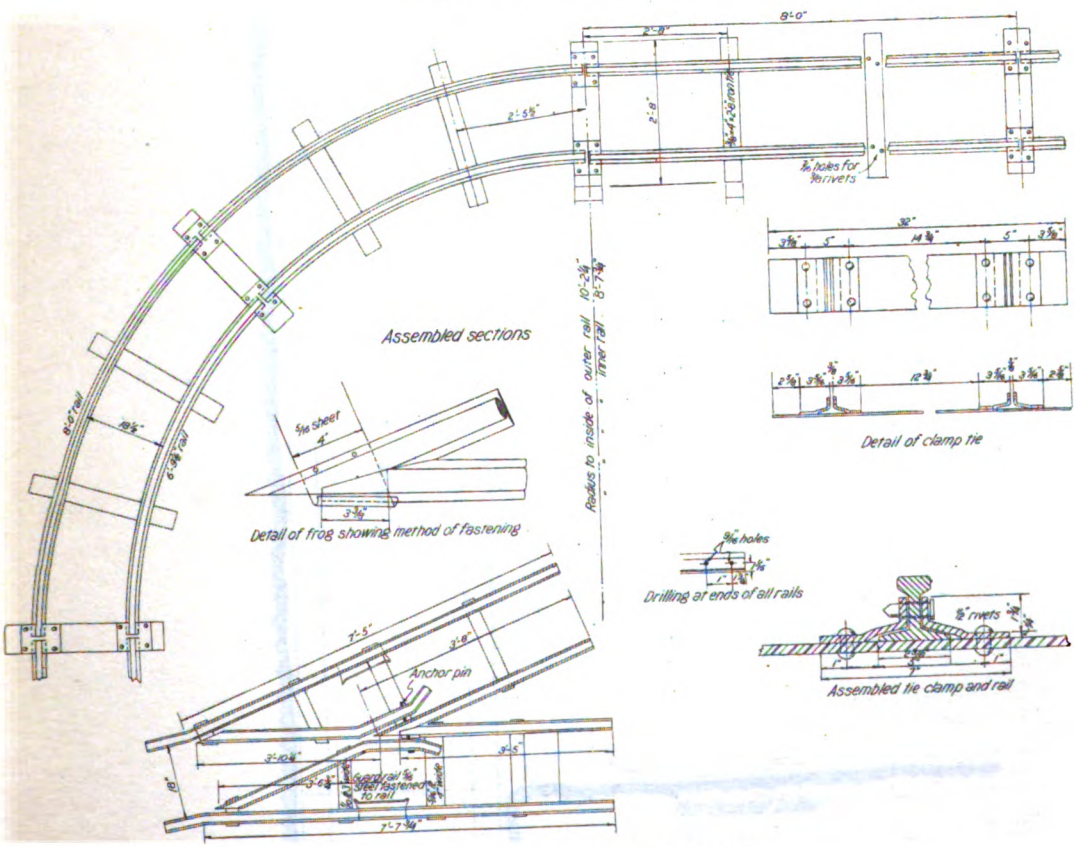
Assembled sections
Detail of frog showing method of fastening
Detail of clamp tie
Drilling at ends of all rails
Assembled tie clamp and rail
Anchor pin

Figure 22.- Sectional stope track

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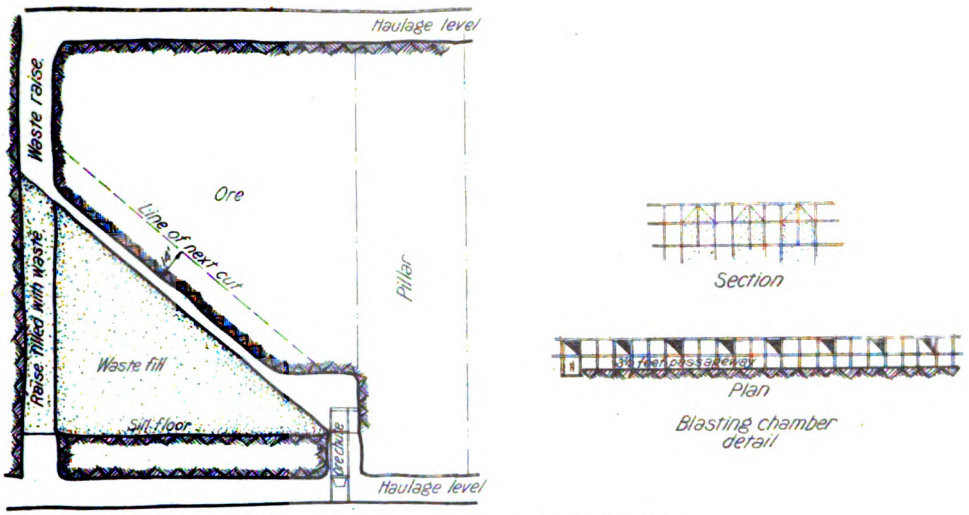

Figure 23.-Typical inclined cut-and-fill stope

Figure 24.-Detail of flooring, inclined cut-and-fill

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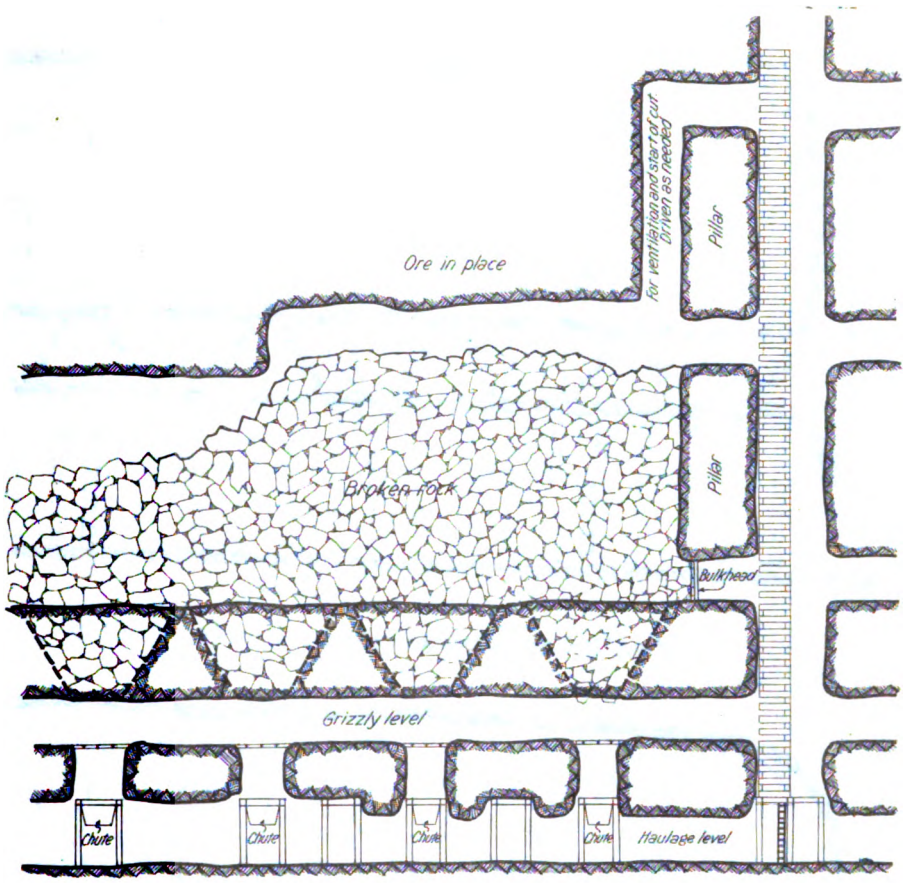

Figure 25.- Shrinkage stope, longitudinal vertical section

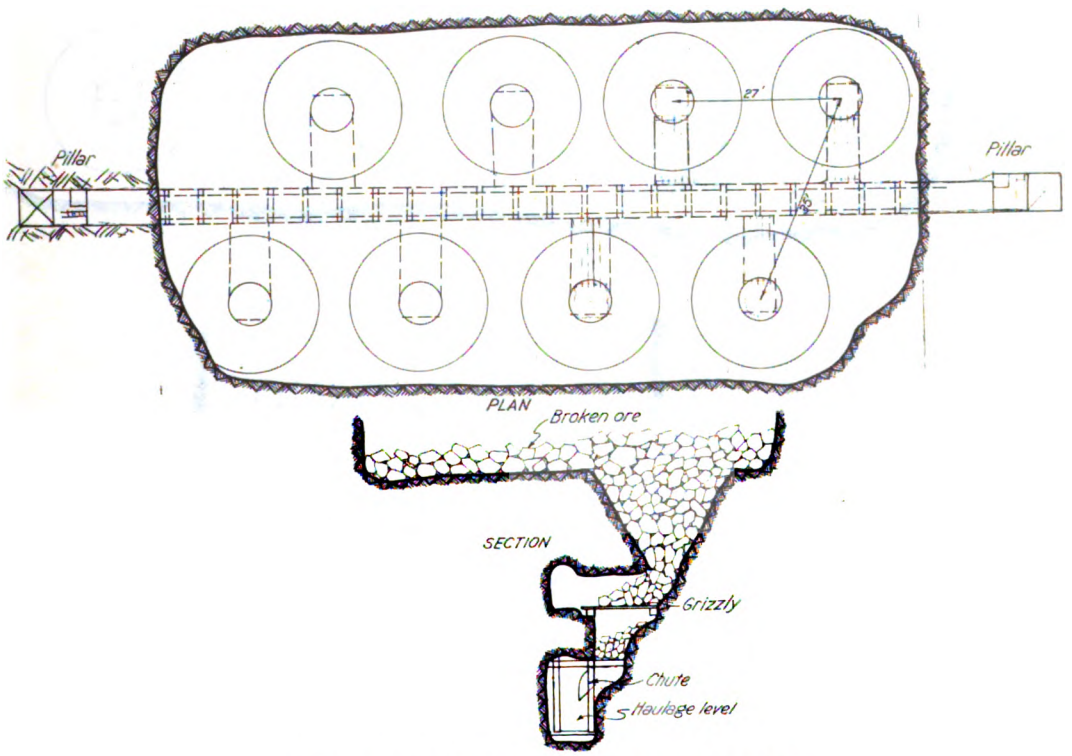

Figure 26.- Shrinkage stope, plan and transverse section

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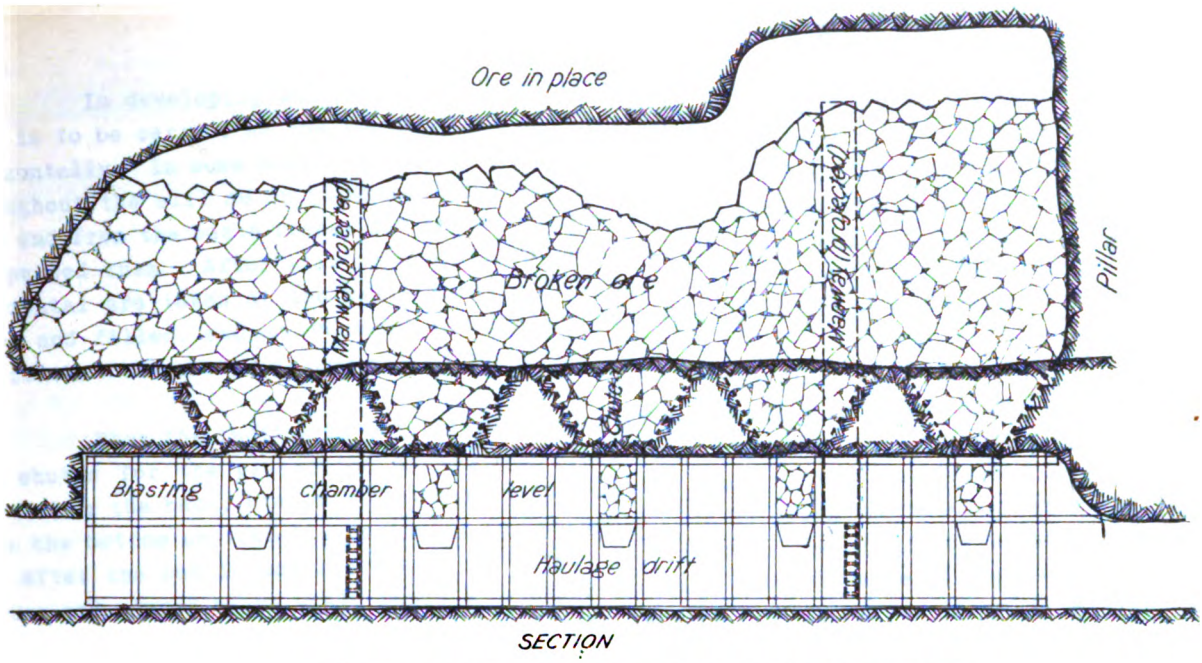
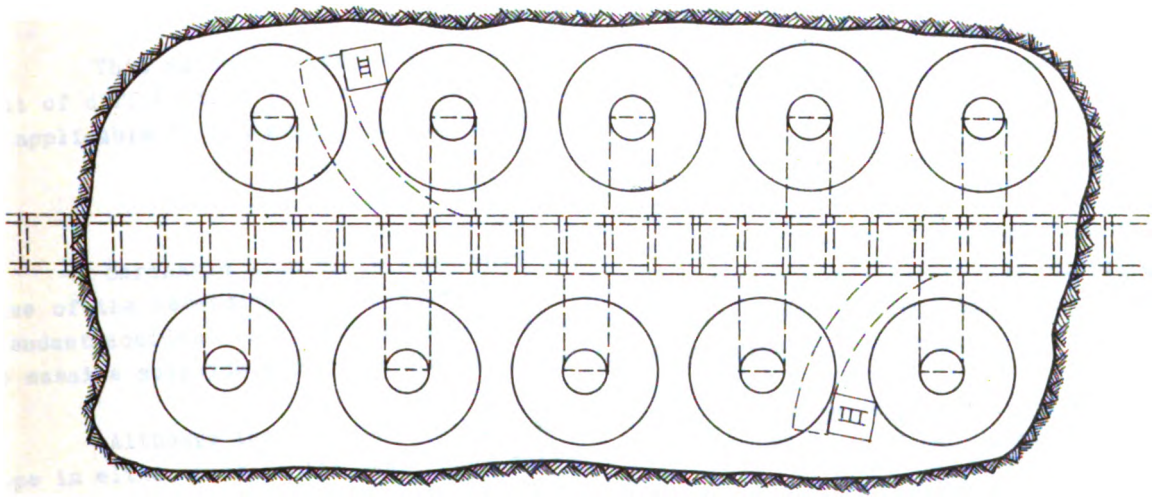


Figure 27.-Shrinkage stope, longitudinal vertical section



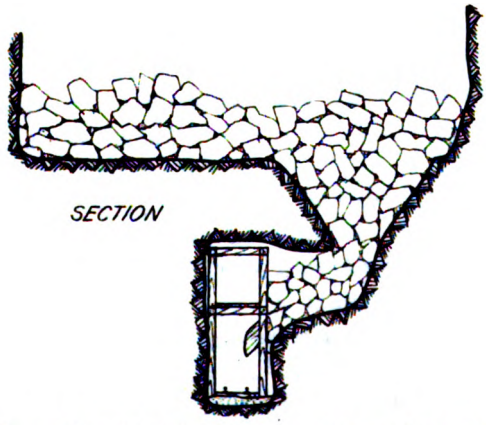


Figure 28.-Shrinkage stope, plan and transverse section

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In developing an inclined cut-and-fill stope, a waste raise is driven at the end that is to be carried as the top of the incline. Then the whole area to be stoped is silled horizontally. In some cases where the stope is to be of considerable size, chutes are placed throughout the sill as in horizontal cut-and-fill. These chutes, except those at the opposite end from the waste raise, are later bulkheaded and abandoned when the toe of the incline has passed them. After silling operations are complete, two cuts inclined at 40° from the horizontal are taken at the waste raise. This is followed by filling, then another cut is mined and filled, and so on until the incline is fully established and the toe of the fill has reached the chutes at the end of the stope.

When the incline is established, a row of square sets is placed along the wall over the chutes for the protection of the shovelers (see fig. 23). A horizontal cut is next taken from the back, starting at the wall and meeting the incline. The incline is then mined from the bottom up, usually beginning in the center and carrying the bench up as an inverted V. After the cut is completed and the broken ore removed, the new floor is started at the bottom and placed at an angle about 3° steeper than the natural angle of repose of the waste fill. Only 5 to 10 feet, measuring along the incline, of floor is placed at a time, and waste fill is introduced at the top of the incline. This method insures a compact fill under the floor and reduces the breakage of flooring to less than 10 per cent (see fig. 24). The fill is introduced through a raise at the top and trammed along the crest in scoop cars.

This method is considerably cheaper than the horizontal cut-and-fill, but on account of difficulties in sorting waste and placing bulkheads under slabs in the back it is not applicable to all ground in this mine.

Shrinkage

Shrinkage stoping is used only to a very limited extent in the main ore body because of the necessity of subsequently building gob fences and filling with waste with an attendant accident risk. Consequently, shrinkage is now used only in isolated ore bodies in the massive sulphide where the standing quality of the ground is excellent.

Although there are two methods of developing shrinkage stopes, the operation of the stope in either case is the same. One method is shown in Figures 25 and 26 and the other in Figures 27 and 28.

In either method, the stope is silled in the same manner as a cut-and-fill stope, and successive 7-foot cuts are mined, starting at the raise, and using flat holes. The small cut is used to avoid burying large bowlders so deeply that they can not be drilled and blasted before being drawn down. Just enough ore is drawn to allow comfortable working room between the broken ore and the back.

After completion of breaking, the back of the stope is carefully barred down and in some cases gunited to insure a safe back until the stope can be drawn empty and filled. During the drawing-off operations, men are kept in the stope to bar down the walls and leave them in safe condition. After the stope is drawn empty the chutes in the bottom are bulkheaded, and the stope is floored in the same manner as cut-and-fill stopes. Filling then begins by dumping directly from the level above. As the filling progresses gob fences are built against any ore walls that may be in the stope.

This method is approximately 10 cents per ton cheaper than cut-and-fill stoping,

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but on account of the high accident risk, particularly during the filling period, it is not used except where conditions are ideal.

Horizontal Square-Set

Square-setting is used in the extraction of the vertical pillars between cut-and-fill stopes, and also in some areas in the main stopes where the ground is heavy. The size of square-set stopes varies considerably. In the vertical pillars the sections are necessarily small, rarely exceeding 35 sets in area, while in the main stopes there may be as many as 250 sets on a floor. The sets are 5 feet 6 inches square by 7 feet 2 inches high. The timber used is all 10 by 10 inch native pine. The shoveling floor is made of 2-inch plank, and the working floor of 3-inch plank held on by 2 by 2 inch lagging strips nailed to the caps and girts.

The operation of the stope follows the usual square-set and fill practice (see fig. 29). Never more than two floors are opened at a time, and the fill is kept as close behind the muckers as possible. The fill is introduced through a raise to the level above, and distributed in the stope by means of narrow end-dump cars running on sectional track with short-radius curves so as to obviate the necessity of turntables or turnsheets in the stope. The narrow car and special track allow plenty of clearance between the car and the posts.

An adaptation to square-set practice of the chute shown in Figure 30 is used to reduce the number of chutes and to shorten the shoveling distance.

Figure 31 shows the method of offsetting a main chute when it becomes necessary due to an offset in the ore limits.

Inclined Square-Set

As shown in Figure 32, the inclined square-set method is used principally in the extraction of level pillars. Its main advantages are an increased tonnage from a given area, and a lower cost per ton as compared with horizontal square-setting. It also has the advantage that there is less weight in the brow, since there is no large square brow at any place. The only serious disadvantage is the difficulty in sorting.

Such a stope is started from a raise in the center of the stope, usually by taking out four rows of sets on the second floor and two on the third before any fill is introduced. After the incline is established, stoping on each new cut is started at the bottom and progresses upward to the crest. One side of the rill is mined while the other side is being filled. The mining procedure is similar to that for any other square-set stope; the only additional precaution necessary is that more care must be exercised in the side blocking to prevent the side thrust of the waste, which is confined to a steeper angle than its natural angle of repose, from causing a side movement, and distortion of the sets. The greatest contributing factor to the success of this stoping method has been the way in which the floor has been placed and the waste introduced. This is shown in Figure 33. The floor is placed at a slightly steeper angle than the angle of repose of the waste, so that when the waste is dumped in at the crest, little or no tamping of waste under the floor is necessary. There are a few voids occasionally under the caps which have to be hand-filled.

Top Slicing

The top-slicing method has been used only in a special case in the extraction of

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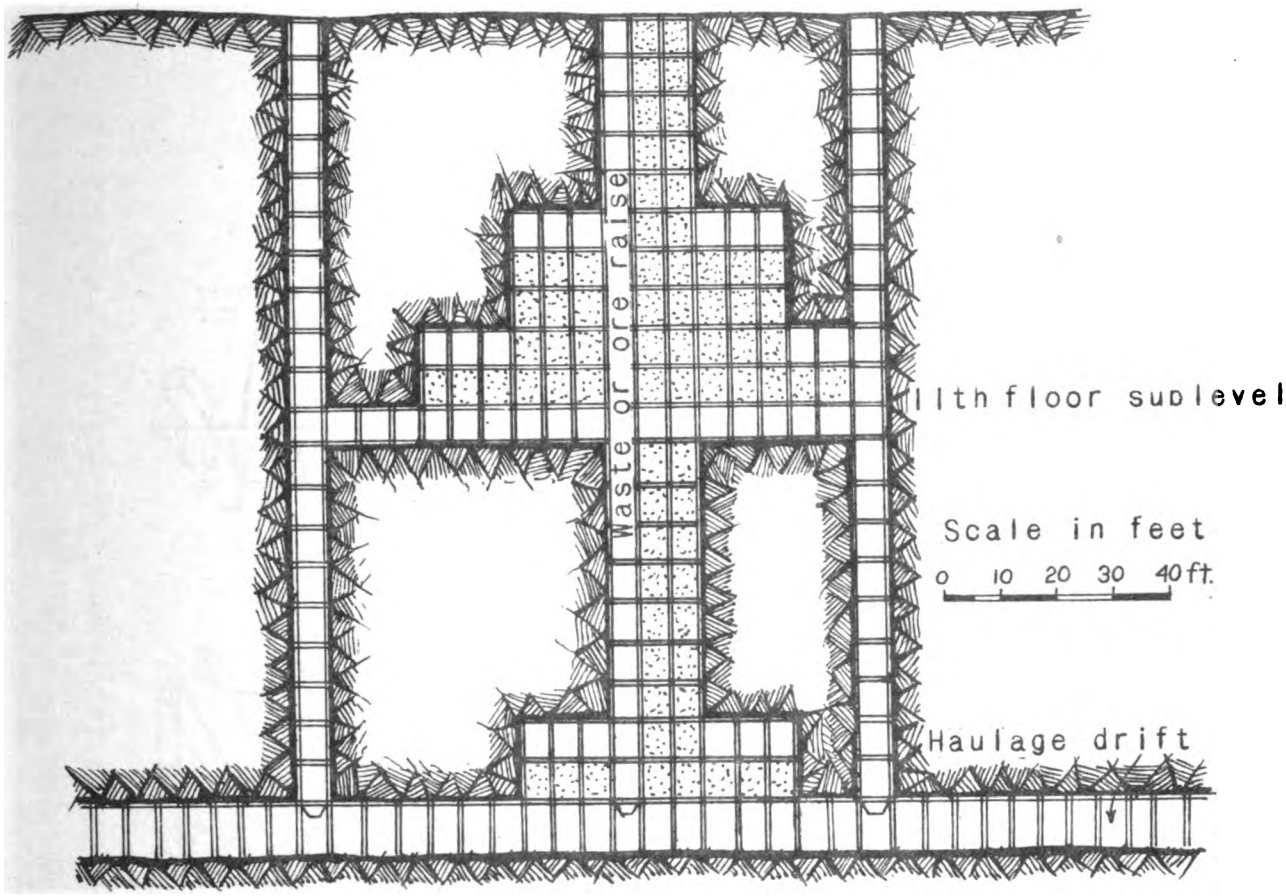
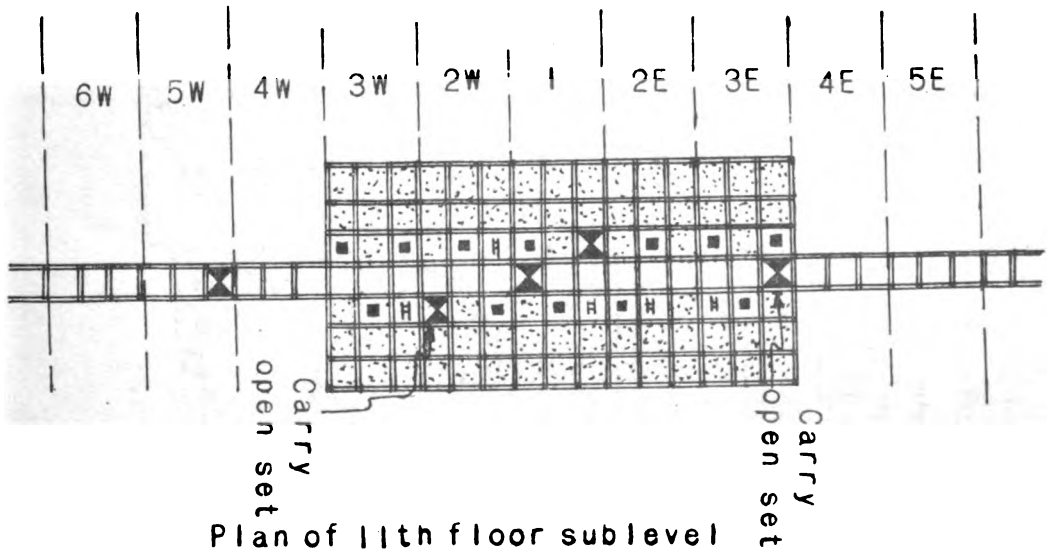


Longitudinal vertical section



Plan of 11th floor sublevel

Figure 29.- Square-set method of mining vertical pillar

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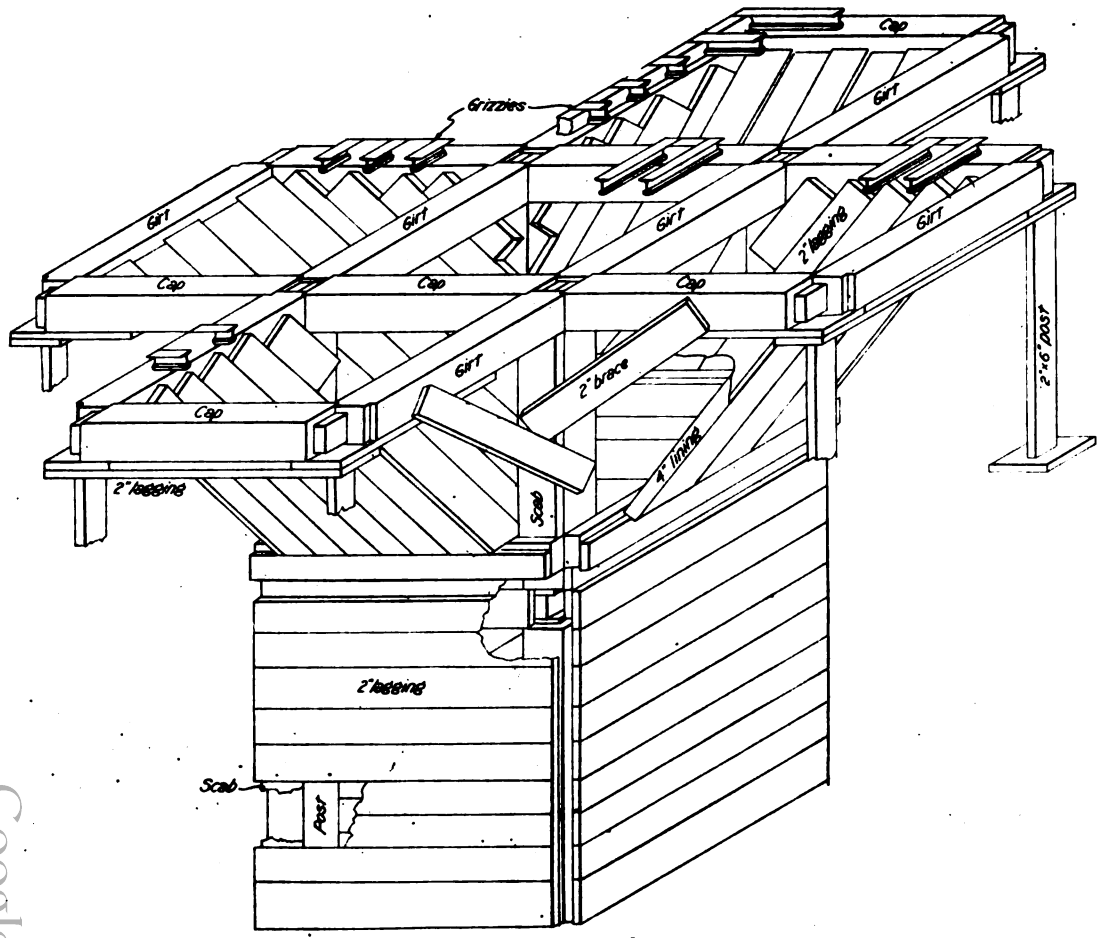

Figure 30.- "Clover leaf" offset chute

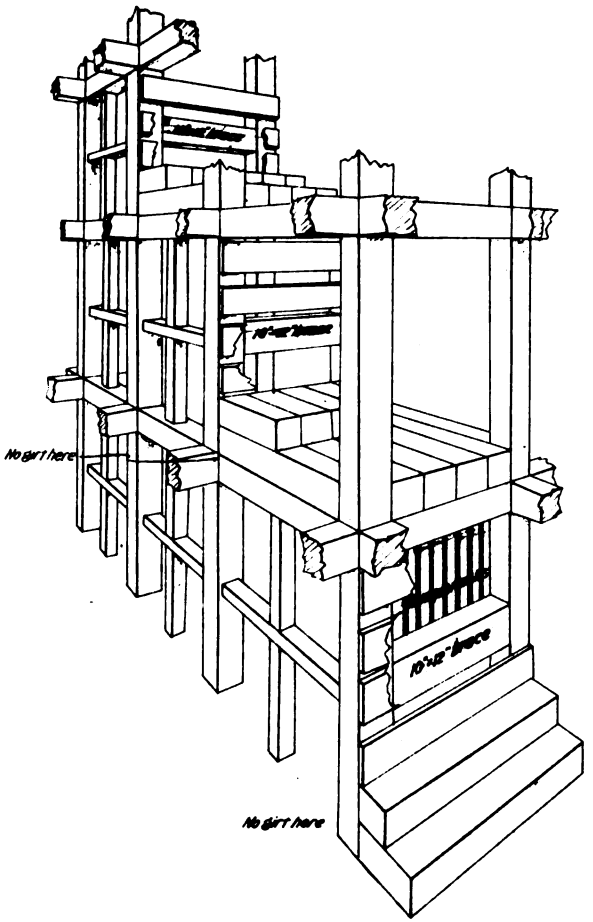

Figure 31.- Offset chute, square-set stope

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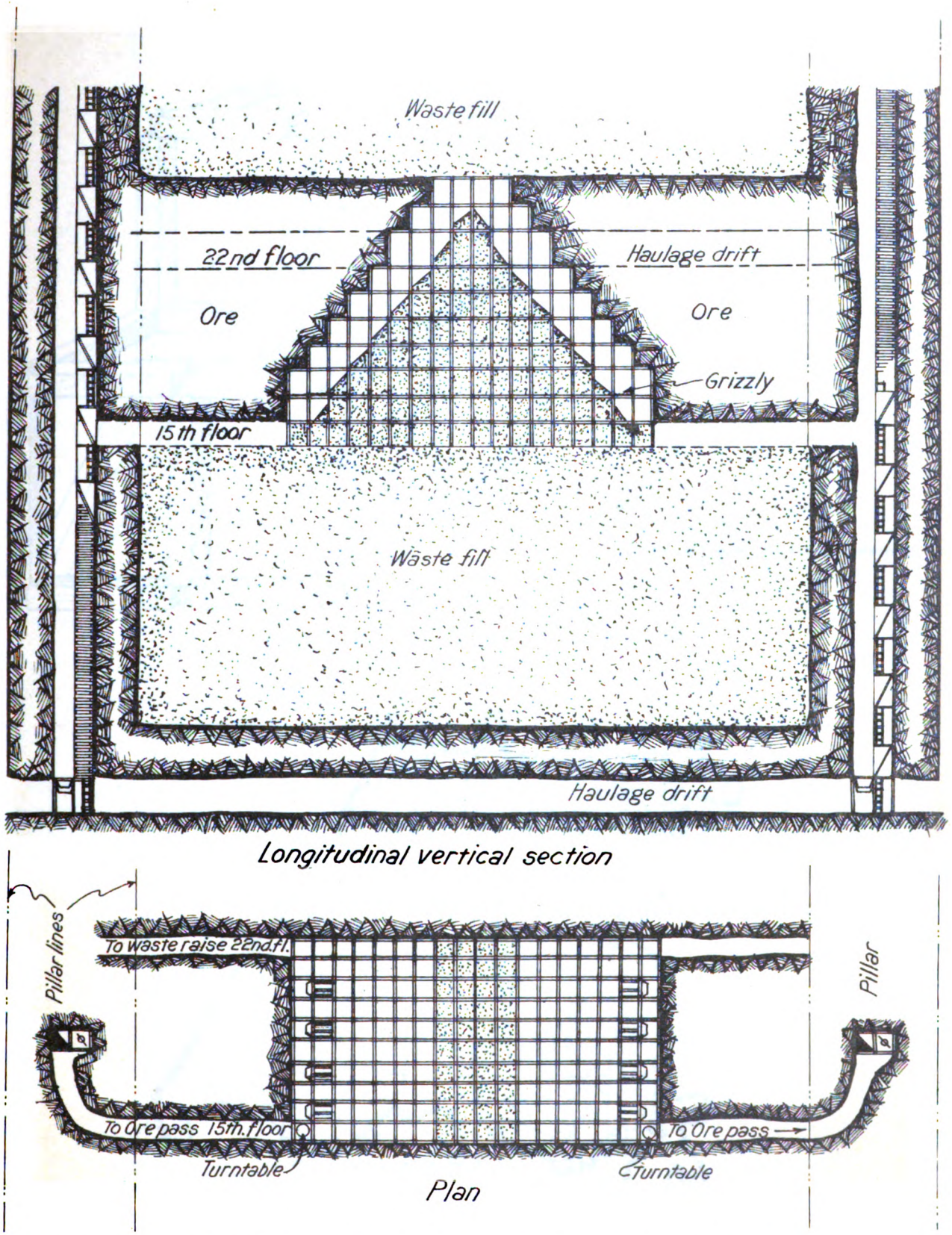

Figure 32.- Inclined square-set method of mining horizontal pillar

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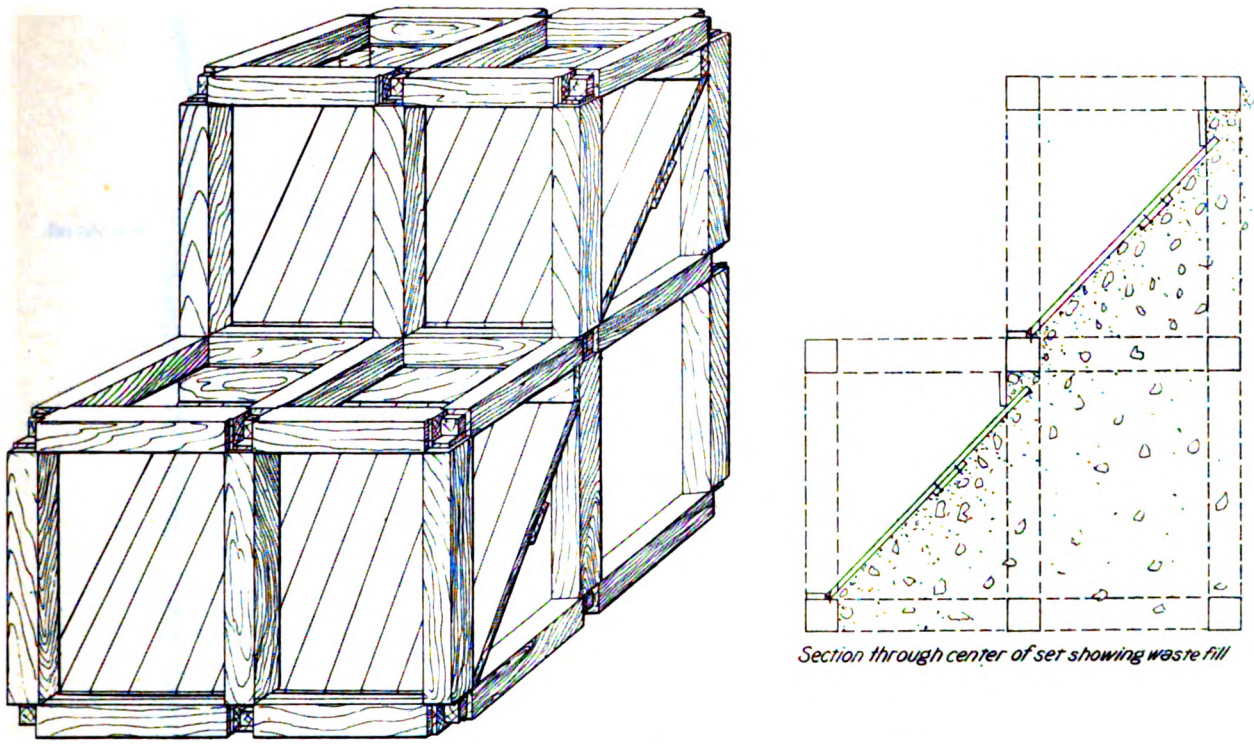

Figure 33.- Method of flooring in inclined square-setting

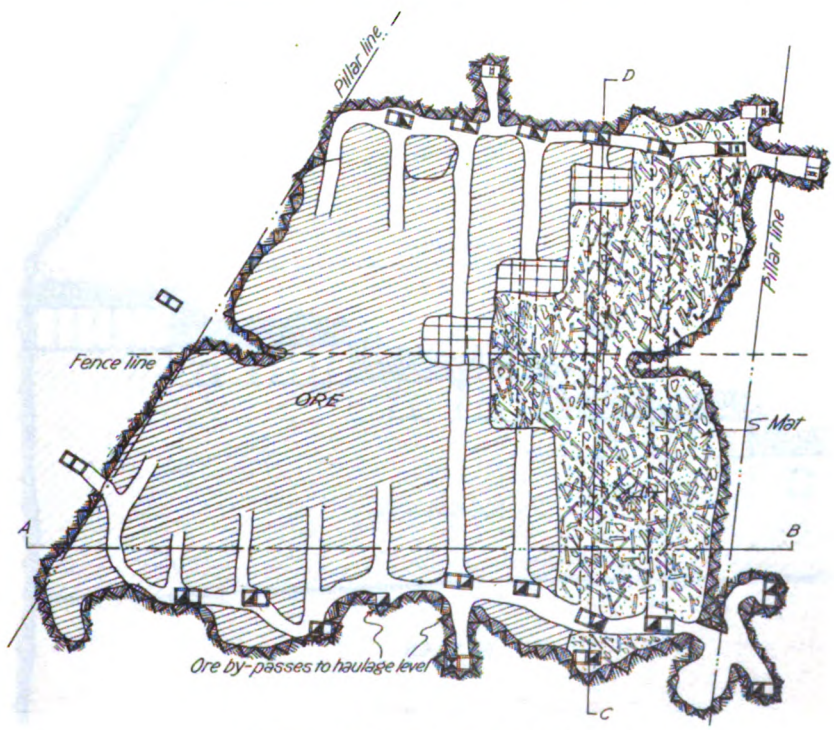

Figure 34.- Plan of No.2 top-slice stope

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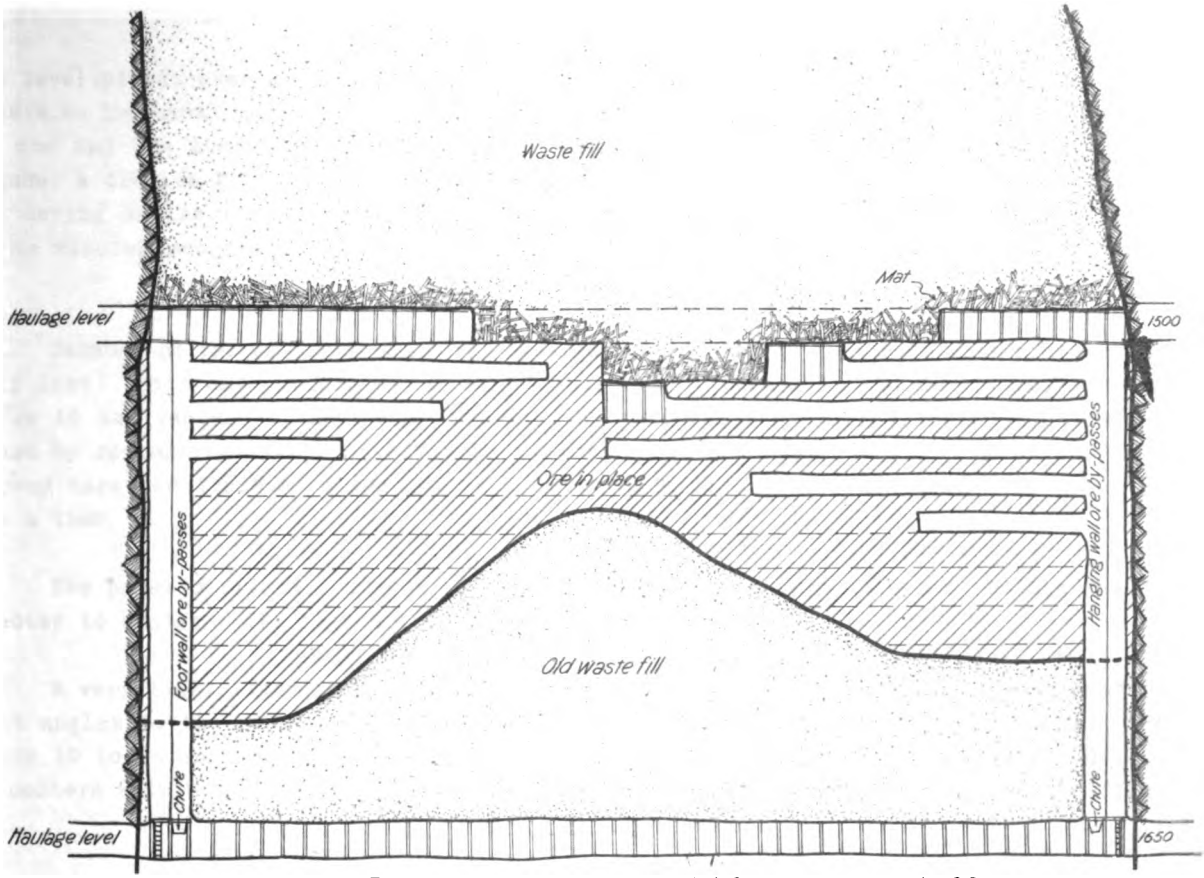

Figure 35.-Vertical section through No.2 top slice stope, section C-D

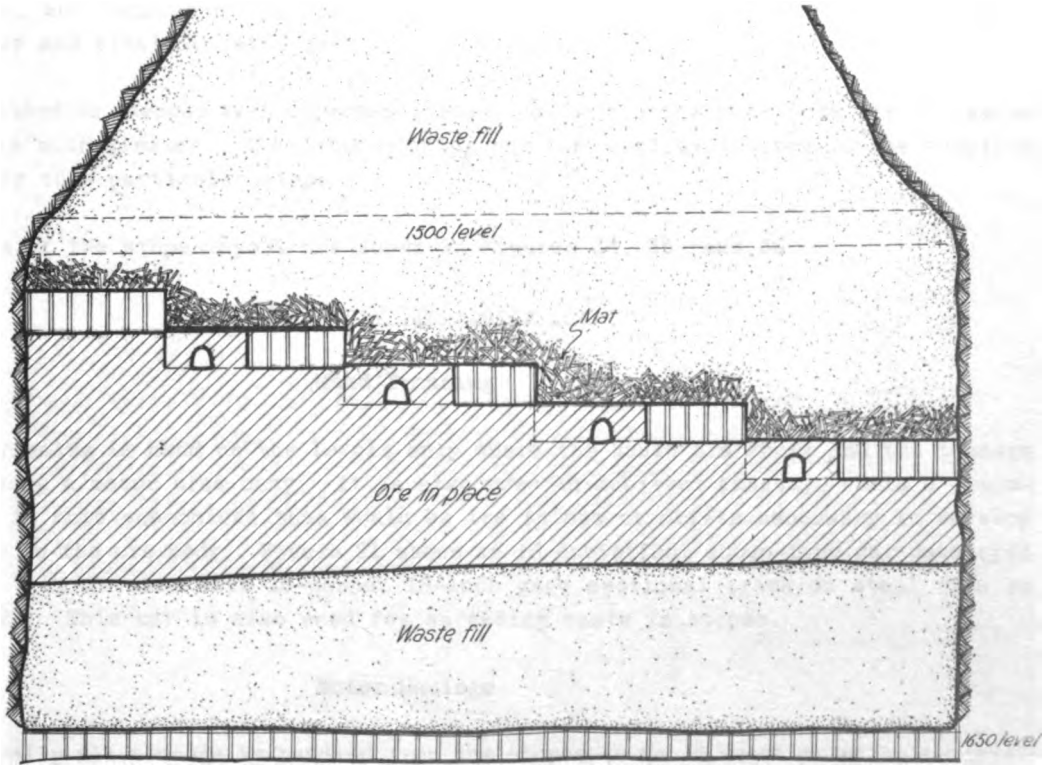

Figure 36.-Vertical section through No.2 top slice stope, section A-B

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a caved level pillar over an old cut-and-fill stope of considerable area. The method is not applicable to the usual conditions at the United Verde on account of the high sulphur content of the ore and its accompanying fire hazard. In the instance noted it was necessary to slice under a diorite fill with very little binding material in it, and at the same time to prevent caving of the hanging wall. For this reason the old stopes above were kept fully filled as slicing progressed, which effectively prevented any major caving of the hanging wall.

Because of the caving of so large an area, all chutes in that area were crushed and entirely lost. This made it necessary to run raises near each wall and drive panel drifts from them to the center of the stope, where actual stoping was started. The stope panels are mined by retreating from the center to the raises on each wall, the ore being shoveled into scoop cars and trammed to the raises. Never more than 25 feet in length of panel is open at a time.

The panel drifts are driven untimbered about 4 by 6 feet in cross section and 35 feet center to center. The slices are 11 feet high.

A very tight floor is laid by placing 4 by 12 inch sills at 5-foot 4-inch centers at right angles to the panel drifts and laying a double 2-inch floor on top of them. The posts are 10 to 14 inch native pine unpeeled poles, and are placed at approximately 5-foot 4-inch centers with 6 by 12 inch head-blocks, and are so placed under the sills of the floor above.

Both primary and secondary blasting of the ore are done with gelatin dynamite and gutta-percha fuse, but because of the fire hazard the slices are shot down with a special permissible powder and electric detonators.

This method is cheaper than square-setting, and the tonnage it is possible to remove in a given time is much greater. The latter factor was the controlling one in the adoption of top slicing for this particular stope.

Details of the stope layout are shown in Figures 34, 35, and 36.

Underground Transportation

Hand Tramming

Hand-tramming is used on the levels only where the hauls are short and the tonnage insufficient to keep a motor crew busy. It is also used in sublevel gangways where a transfer of stope ore is more economical than would be the raises or drifts necessary to develop irregular offsets in the ore body. Figure 21 shows an 18 cubic foot scoop-type car developed for this purpose. It is used with 16-pound, 18-inch gage sectional track on steel ties as shown in Figure 22. This car is also used for spreading waste in stopes.

Motor Haulage

Practically all tonnage is handled from the stopes to No. 5 shaft by motor haulage. The length of haul averages 1,000 feet. Trolley locomotives have been largely replaced by 4 and 6 ton storage-battery locomotives, which handle sixteen to eighteen 30-cubic foot rocker-bottom cars. These cars weigh 2,500 pounds, and are equipped with 12-inch roller-

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bearing wheels. A particular feature of this car is a double trunnion, which gives a minimum overhang in the dumping position and permits dumping in the standard width gangways when handling waste for stope fill. The car is dumped by means of a piece of drill steel, one end of which is inserted in a shallow hole in the drift wall, the other end being placed against a lug on the side of the car. As the car moves forward, the drill steel forces the car over. Several mechanical dumping devices have been tried, but the drill-steel method is simple and effective.

Each level is equipped with a charging station with steel platforms on either side of the track. The tops of these platforms are the same height as the frame of the motor, which permits the battery boxes to be rolled on or off the motors as required. Each motor has an extra set of batteries which is on charge while the other is in service. The charging sets are of the constant-potential type and operate from 250-volt direct-current circuit.

Figure 37 shows a standard chute pocket for ore-passes from stopes. The steel gate is operated by hand, using an 18-inch forged handle with a slot to slip over the truss strap on the gate.

Track

Forty-pound rail laid at 18-inch gage on 6 by 6 by 30 inch ties, spaced at 24-inch centers, is standard for all motor-haulage drifts. Sixteen-pound track, either sectional or laid on 4 by 6 inch ties, is used for hand-tramming. Nos. 4 and 5 manganese steel frogs are standard equipment, and are rebuilt by electric welding when worn down. Switch throws, points, and the necessary rail clamps, spacers, etc., are constructed in the mine shops. A minimum radius of 50 feet is maintained for all motor-haulage track.

Hopewell Tunnel Transportation

All ores from the pit and underground operations, as well as excess waste from the pit, are transported from the main loading bins on the 1000 level through the Hopewell Tunnel to the outside storage bins, which have a capacity of 10,000 tons. Because of the character of the heavy sulphides, particularly the fine calcined material from the fire zone, it is essential that the chute openings be as large as possible to prevent the fines from building up in the bins. Figure 39 shows the general plan of these loading bins, which are very rugged in construction. The gates are operated by air cylinders.

The ore is handled by 40-ton standard-gage cars, provided with compressed-air cylinders for operating the dumping mechanism. Twenty-five-ton trolley locomotives, equipped with two 75-horsepower commutating pole type motors and automatic air brakes, handle a 9-car train at a speed of 7 miles per hour. The average gross weight of the train is 540 tons.

With four motor crews of three men each, 5,000 tons of ore daily are transported over this 1.8-mile haul. The track is standard gage, with 75-pound rail, and has a grade of 0.5 per cent in favor of the load.

Hoisting Practice

Because of the abrasive effect of the hard sulphide ore, all chute, grizzly, and pocket construction is very heavy. Loading pockets (fig. 30), are of 500-ton capacity, and are arranged according to general practice in the Southwest. The upper gate is of the undercut plate type, while the measuring chute has an undercut arc gate with a movable baffle

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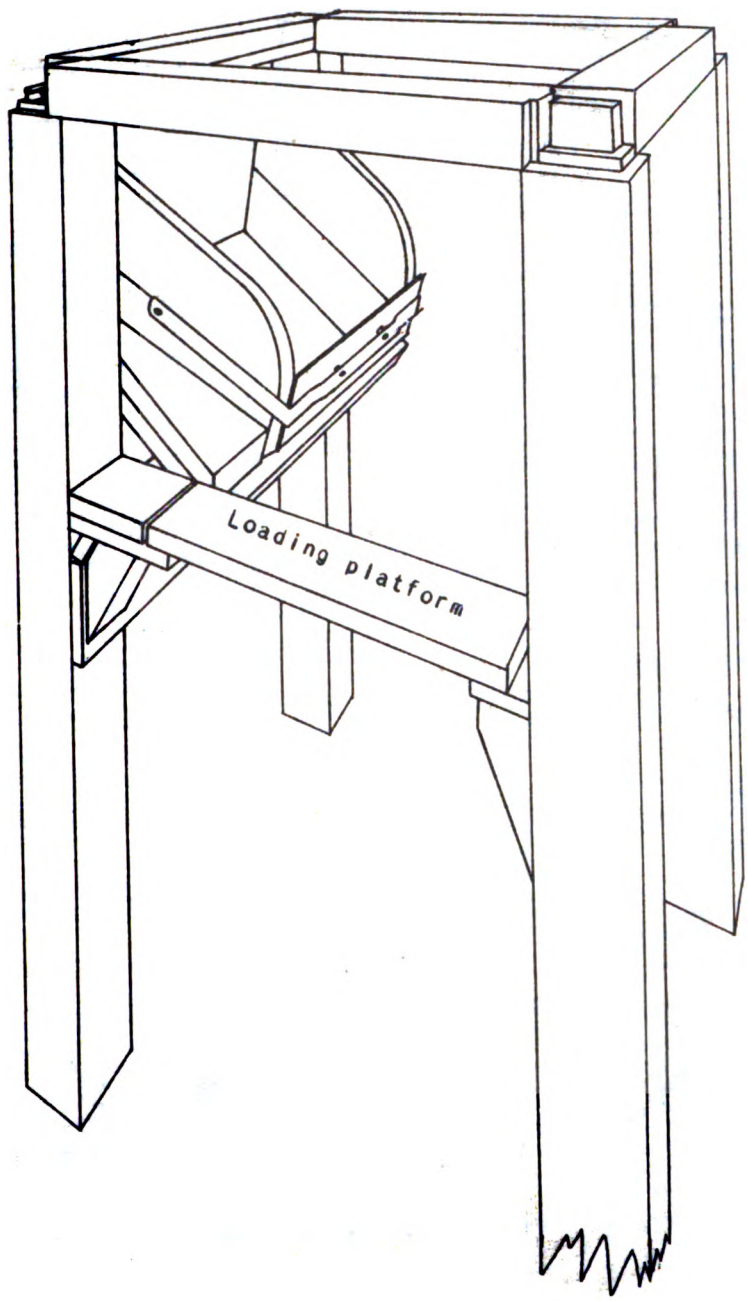

Figure 37.- Standard loading chute

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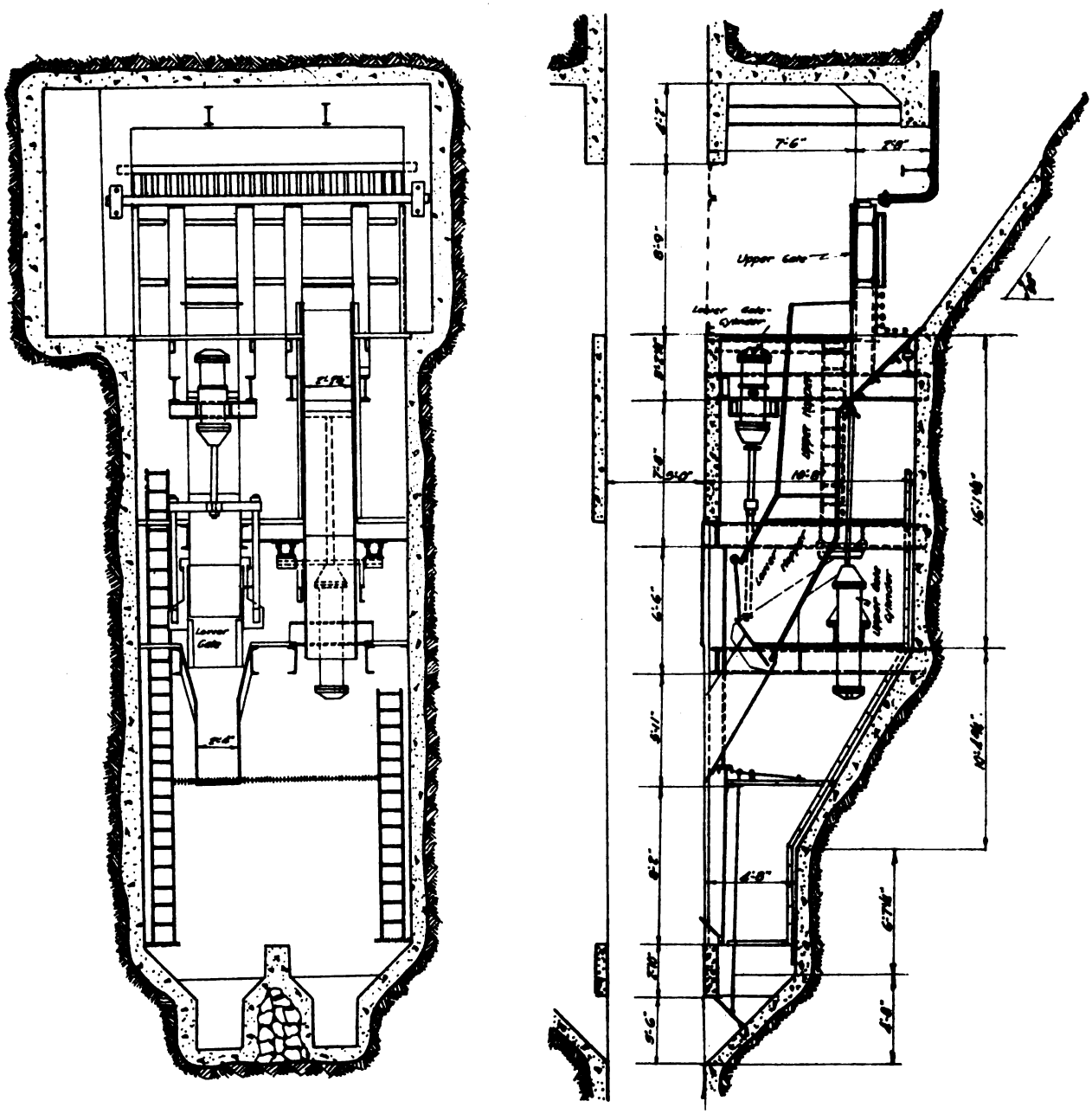

Figure 38.- No. 5 shaft skip-loading pocket

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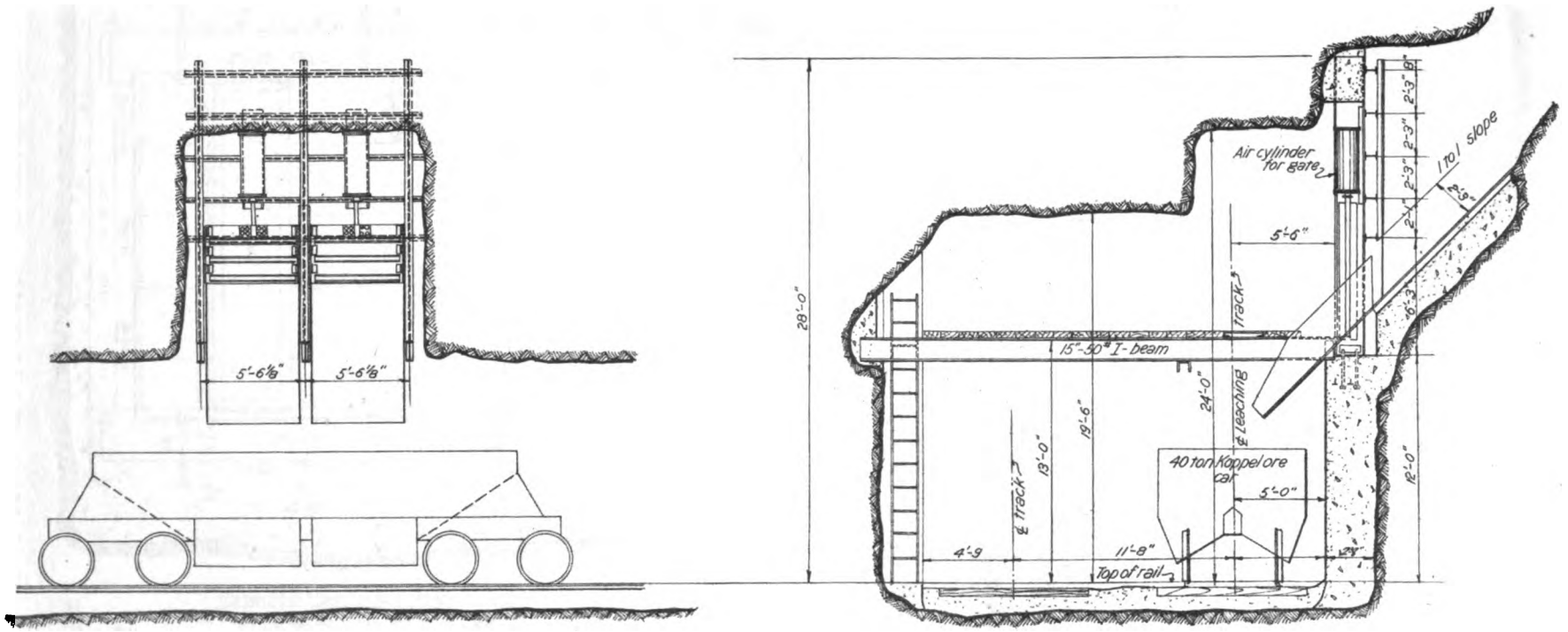

Figure 39.- Hopewell tunnel loading bins

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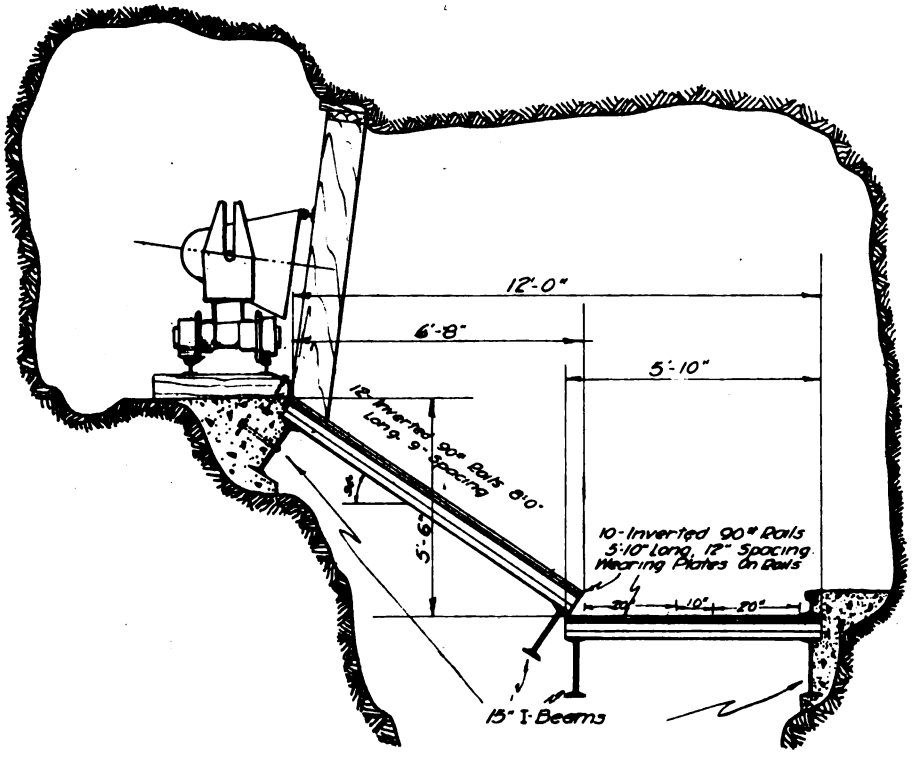

Figure 40.- Standard ore-pocket grizzly

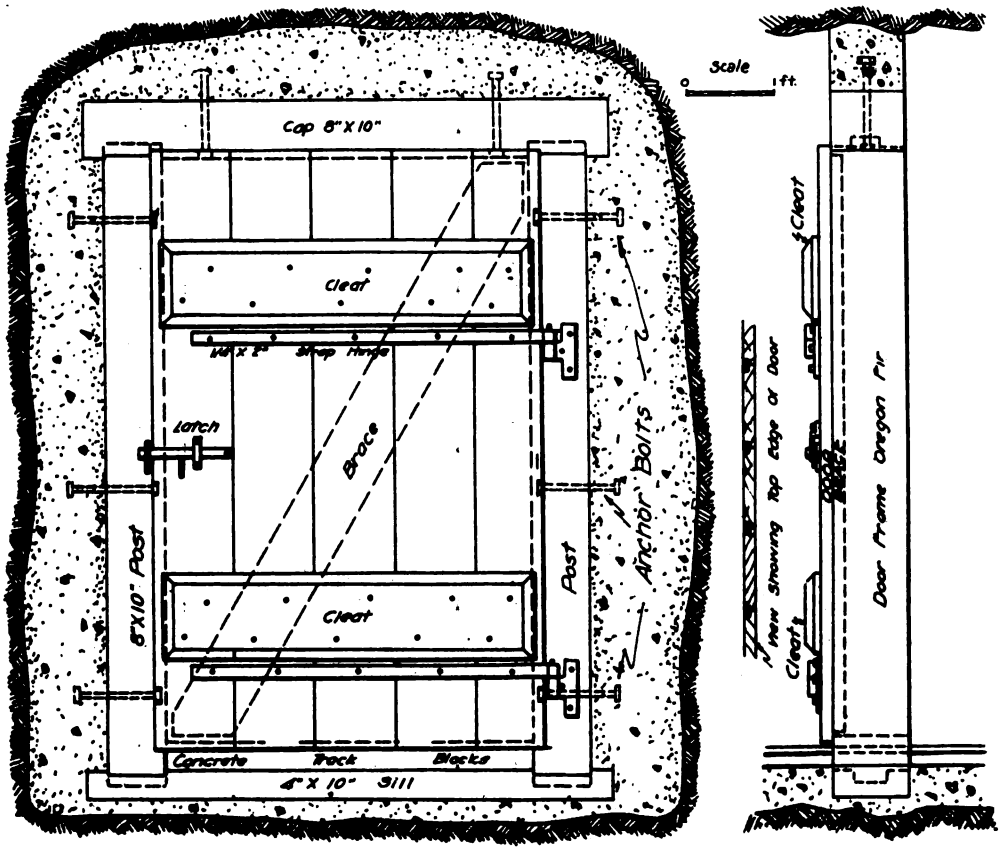

Figure 41.- Standard mine motor door

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front. Both gates are operated by compressed air. The average loading time is 7 seconds. The valve on the air line leading into the pocket is provided with a padlock, so that no one but the skip loaders can operate the gates.

Figure 40 shows the grizzlies above each pocket. The grizzly bars are inverted 90-pound rails which have been annealed and faced with a 1 by 5 inch wearing plate. The inclined rails are spaced to give 9-inch openings, and the lower horizontal rails have 12 by 20 inch openings. This construction permits an entire train to be dumped without stopping to bulldoze the bowlders. Each level is equipped with two sets of grizzlies, with transfer raises to the level or loading pocket below. This is essential, as mill ore consisting of schist and porphyry, and direct smelting ore consisting of massive sulphide and high-grade-schist (over 7.0 per cent copper), are handled separately.

The spill from hoisting 2,000 tons daily averages 6 tons. This spill being abrasive, wears the shaft walls, and it has been necessary to build up the curtain walls by means of gunite.

A heavy hinged bulkhead, or deflector, constructed in the hoisting compartment, deflects the greater portion of the spill into the ore pockets. This deflector, located at the 1950 level, is electrically operated by remote control from the engineer's platform, and pilot lights indicate the open and closed positions. The portion of the spill which reaches the bottom of the shaft is drawn through standard chutes into an 18 cubic foot car and hoisted through the manway compartment to the level above.

No. 5 hoist station is located on the 1000 level. It is 47 by 81 feet in plan and 22 feet in height, and is lined with reinforced concrete. The cableway extends at an angle of 60° to the headframe on the 800 level. The sheaves are 10 feet in diameter, and are of the plate-type construction with case-steel liners.

The ore hoist is of the double-drum, single-reduction, geared type, driven by a 650-horsepower, direct-current, 500-volt motor. Power is supplied by a 600-kilowatt generator set with a 20-ton flywheel. Each drum is 10 feet in diameter with a 5-foot face, and holds 2,500 feet of 1-3/8-inch, 6 by 19 hoisting rope in two layers. The hoisting capacity is 260 tons per hour at a rope speed of 890 feet per minute. The total load on the cable is 15 tons. The average life of a cable is two years and the ore or waste hoisted per rope is 760,000 tons.

The skips, which operate in balance, are of 112 cubic foot capacity, equivalent to 8 tons of sulphide ore, or 6 tons of schist ore. They are of rugged construction, have 3/4-inch plate liners, and weigh 7 tons each.

The ore is hoisted to the 800 level, where it is dumped through an electrically operated "selector" to one of three storage bins, depending on the class of ore or waste being hoisted. This selector is operated from the hoistman's platform, and colored pilot lights indicate its position.

The hoisting crew consists of a hoistman, skip loader, and helper.

BONUS SYSTEM

A fair amount of work done safely by an average man in a given unit of time is the

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standard for any particular job. The efficiency of the job is calculated as follows:

If a standard for a job is one unit per shift, and a man accomplished one and one-half units per shift, he is rated as being 150 per cent on that job. He receives as bonus one-half of his increase in efficiency over 100 per cent, or 25 per cent of his day's pay rate. The company benefits by the other half. Putting this into a formula, it may be expressed as follows:

$$T_2 = P/S;\ E = T_2/T;\ B = 1/2(E - 100)\ W$$

S = Standard (unit of work per unit of time)
P = Work accomplished
T = Time required for doing P
T = Time allowed for doing P according to S
E = Efficiency in per cent
W = Wage rate per unit of time
B = Bonus paid

Obviously, the major problem is to determine correct standards of work. The setting of standards depends largely upon past performance and upon the judgment of the man setting the standard. The men chosen to set the rates must have had considerable experience themselves in doing and in bossing the work they are rating. Such a man inspires the respect and confidence of the workers. The foremen are consulted whenever a new standard is to be set.

Since 1924, a good deal of work in the mine has been paid for on a contract system. practically all of the new timber work, drifting, crosscutting, and raising is done on contract. However, what we call a contract is not a contract in the true sense of the word, because the agreement is verbal and the man assumes no obligation. The company agrees to pay a certain price for a certain unit of work, but the man is guaranteed day's pay no matter how much he accomplishes. Also, the company retains the right of complete supervision over the work and the manner in which it is done. The contractor has nothing to say about who his coworkers or helpers will be, though, of course, an attempt is made to put men together who have equal ability and who will work together to advantage. Each man is responsible to the company only and they are paid separately by the company. Two examples of contracts follow:

A.

Credit 10 chutes @ $10 =	$100.00	Deduct 5 shifts @ $5.23 =	$26.15
	49.55	5 shifts @ 4.68 =	23.40
Due contract	50.45	Total deductions	$49.55

Due timberman in addition to day's pay: 53 per cent of $50.45 = $26.75
Due helper in addition to day's pay: 47 per cent of $50.45 = $23.70

Division of the profit is made according to the ratio of the day's pay of the two men.

B. A drift is driven under contract at a price of $5 per foot. This price includes mining labor and explosives only, as the mucking is let

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separately. The miner drives the drift 100 feet in 20 shifts and uses 30 boxes of powder.

Credit 100 feet @ $5.00	$500.00	Deduct 20 shifts @ $4.95	$99.00
	339.00	30 boxes powder @ $8	240.00
Due contract	$161.00	Total deductions	$339.00

Miner receives $161.00 plus day's pay for 20 shifts.

Two advantages of this type of contract over the bonus system in use here are that the contract is easily understood by the miner and he visualizes his units of work accomplished in dollars, which acts as an incentive to greater industry. Also, in the case of development work, it rewards the miner for saving explosives, while under the bonus system the tendency is to be extravagant and wasteful of explosives.

This type of contract has been very successful in lowering unit costs and in increasing the earnings of the men. It has been tried to a limited extent in mining in stopes and within the next year will be given a thorough tryout for all extraction operations.

A great deal of care must be taken to set a correct contract price in this method of payment because the company does not participate in the bonus, and if the price is set too high the unit cost to the company will be too high.

A great many of our contract prices have been arrived at from the bonus standards, in conjunction with the average bonus earned over a long period of time on a given class of work. For instance, the price of a drift would be set as follows:

The bonus standard for the drift might be 2 feet per shift. The average bonus for drift work was somewhat under 50 per cent, but good miners, under favorable conditions, often made 50 per cent or better. To make 50 per cent bonus in the above drift, the miner would have to average 4 feet advance per day, and this would cost the company $4.95 plus 50 per cent of $4.95, equalling $7.42, or $1.855 per foot for labor. Then, if $1.90 per foot were allowed for explosives, the contract price would be $3.75 per foot. If the miner broke 4 feet per shift and just used the allowed amount of powder, he would make just the same amount of money as if he worked under the bonus system. To make more money on the contract than on bonus, he would have to reduce the amount of powder used per foot or increase the amount of advance per shift. If he fails to make an advance which would have paid 50 per cent bonus under the bonus system, or if he uses an excessive amount of powder, he will not make as much as if he were on bonus instead of contract.

The company gets the work done at a certain fixed price whether the miner makes a good bonus or not, unless he shows a loss. In case of a loss by the contractor, the cost to the company is increased by the amount of the loss over the agreed price.

If the contractor is not doing reasonably well, an investigation is made of the conditions governing the operation, and the contract price is changed if unforeseen difficulties have arisen. If, however, the fault is found to be with the contractor's ability, he is either transferred to some work he is better fitted to accomplish or he is discharged.

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The contract price is subject to change at any time if conditions over which the contract has no control are changed. For instance, the formation in a drift may change from soft to hard rock, in which case a new price would be set starting at the contract. If an error, due to misjudgment, has been made in setting the contract price, the change in price is retroactive and starts when the work started if the original price was too low. If the price was set too high, the cut does not take effect until the first of the month following the time the mistake was discovered. Although this gives the men an advantage over the company, it tends to inspire their confidence in the system.

Most of the underground work and all of the mechanical and open-pit work is still under the bonus system. Various types of standards or allowances are set for the different operations according to our ideas of what seems to be the most equitable method of settlement.

Type of work	Unit of standard
Underground work:	
Mining in drifts and crosscuts	Feet per shift advance
Mining in stopes	Cubic feet per shift broken
Laying track, hanging pipe, etc.	Feet per shift installed
Hoisting ore	Skips per shift hoisted
Hauling ore	Cars per shift hauled
Mucking	Cars per shift filled
Open-pit work:	
Trucking ore	Tons hauled per shift
Electric shovel operation	Cubic yards dug per shift
Truck maintenance	Dollars allowance per truck shift
Mechanical work:	
Machine work	Given number of hours allowed per unit of work
Sharpening drill steel	Time allowance per rock-drill shift

The above are only a few examples of the many operations, but they are representative. The drill steel sharpening standard is not based on the number of pieces of steel sharpened because when this method of payment was tried, the operators burned the steel in an effort to increase their production. When burned steel went underground, the bit soon failed and the steel was out again for resharpening. This tended to increase the work in the sharpening shop, and thus the men were paid a premium for doing poor work. Under the present system, when perfect steel is sent underground, a minimum amount of resharpening is required, the shop crew is as small as possible, and a maximum of bonus is made by the men, so that the premium is now paid for good and efficient work. The same method of payment is used in the rock-drill repair shop. They are allowed so much time per drill in operation, so that drills sent down in perfect order will stay down the longest time possible and reduce the work in the shop to a minimum, thus allowing the men a maximum of bonus. This method of reward does very well for maintenance and repair work where certain equipment is in constant use and requires constant attention.

The organization of the bonus department consists of a chief bonus engineer, an assistant chief bonus engineer, an office clerk who posts the time of the workmen on the various job sheets, four bonus engineers who handle the underground work, and two bonus engineers who look after the outside and mechanical work.

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The engineers take the cubic feet broken, explosives used, cars trammed, etc., from the shift bosses' distribution sheets each day. They are constantly visiting the working places, so that they are fully acquainted with the details of the work going on. Each engineer computes the efficiency of all the jobs under his jurisdiction and the computations are then checked by the head of the department or the assistant.

The number of men required to carry on bonus work depends largely on the number of jobs. In turn, the number of jobs depends on the amount of attention which it is thought advisable to spend on the details of each major operation. For example, the shops may be given the job of completely overhauling a Mallet locomotive. A standard of 500 shifts might be set for the total job. This would be one job and would require very little work from the bonus engineer after the standard was set. The job might be divided into machine work, boilermaker work, steam fitting, and electrical work, making four jobs out of one, which would require practically four times as much work in the bonus department. Or, each individual operation on the overhaul job might be considered as a separate job, rates set, time kept, and the computations made on a great many details which would take considerable time on the part of the engineer. This last method of handling the bonus is the most accurate and is the method employed where the results obtained justify the cost of the clerical and engineering work. In some cases, it would cost as much to obtain an accurate standard on a particular job as it would cost to do the work. This is particularly true for work which is done once and is not repeated. In such a case no attempt is made to set a standard for the job. If the men doing such work have been regularly working on bonus, they are paid a flat rate of bonus for the time spent on the job. This does not furnish any incentive for faster work, but it prevents dissatisfaction on the part of the men required to do the work, and it is cheaper than spending the time to obtain a standard.

Some of the regular work about the plant is of such a nature that it is impractical to put it on a bonus basis, and such work is done on "company time," no bonus being paid over the daily wage rate. If a man works part of a shift on a bonus job and part of a shift on a company time job, he will be inclined to charge most of his time to the company time job, so as to increase his bonus. The boss must therefore keep the men's time carefully and accurately.

The bonus is figured on the first of each month for the work done the previous month, and the money earned is paid on the check the men receive on the 21st of the month. In case a man quits during the month, his bonus is mailed to him immediately after the bonus pay day.

We have found that in every case where a fair standard has been set for bonus work, it has resulted in lowering the cost of that operation, and that the men doing the work earned more money than before.

VENTILATION, MINE FIRES, AND SAFETY METHODS

Ventilation

The United Verde mine is mechanically ventilated throughout. This is necessary because of the poisonous sulphur fumes which are at times generated when blasting in the so-called "massive sulphides." It has also been found that control of air currents is sacrificed to some extent when natural and mechanical ventilation are combined. Positive control as to direction and volume is considered very important normally and absolutely necessary in case of a mine fire.

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The virgin rock temperature on the 3000-foot level was 95°. Temperatures taken below this horizon in diamond-drill holes indicate a virgin rock temperature increasing about 1° for each 150-foot increase in depth.

The mine is ventilated by a main fan on the 1000 level. The air is drawn through an intake air shaft 13 by 13 feet in the clear and 980 feet deep. On the discharge side the air passes through a short duct which contains a multiple-blade damper for volume control, then down an inclined raise to the fresh-air shaft which extends downward to the 3000 level (see fig. 42). This shaft has an effective cross-sectional area of 80 square feet. Fresh air is taken off on all levels from the 1000 to the 3000 level. The volume is controlled on all levels by regulator doors which can be set for any desired volume.

The main return airway consists of a series of raises and connecting drifts of various lengths, beginning on the 2100 level and extending upward to the 900 level. At this point it connects with a shaft which extends to the 300 level. A drift to the surface completes the return air course. This course has an effective cross-sectional area of 120 square feet. Both the main fresh air course and the main return are free of timber or other combustibles throughout their entire length.

The main fan is of the double-inlet, reversible type, with forward curved blades. The rotor is 4 feet 6 inches wide by 9 feet 10 inches in diameter. It is direct-connected to a 400-hp., 2,200-volt, 280-r.p.m., 3-phase, 60-cycle, slip-ring motor. A 350-horsepower, 2,200-volt, 3-phase, 60-cycle, 350-r.p.m., wound-rotor induction motor is used as an auxiliary drive. This motor is belt-connected and can be called into service on a moment's notice by means of a clutch on the fan drive-shaft. Both motor controls are on one switchboard. A push button switch cuts the power from one motor to the other. The speed of both motors can be changed at will by means of a push-button type control, with a bank of resistances in the secondary circuit arranged to give three speeds. The multiple-blade damper was installed to give still greater volume control than is possible with the three motor speeds.

To obtain the maximum benefits from a ventilation system, it must be so arranged that there is a minimum of loss through short circuiting or leakage. Figure 41 shows a standard mine door on a motor level. In ventilating the United Verde mine, a constant check is kept on air currents to prevent loss of air into channels in which it becomes ineffective. Thus unavoidable losses in hoisting shafts are utilized on levels in the upper part of the mine for cooling purposes. Abandoned workings are sealed off in order to eliminate losses or air. Such economies in air are felt to be necessary if the man at the face is to derive the maximum benefits from the power expended at the fan.

No particular attention has been paid to the reduction of the mine resistance by smooth-lining main airways, but all new airways established in recent years have been made larger in order to take care of the resistance factor. A standard cross-sectional area of 120 square feet has been adopted.

In the course of mining it was deemed advisable to leave level pillars in certain sections of the mine. These pillars were later made use of as horizontal fire breaks. All through raises in these pillars were provided with iron doors hung in concrete frames (see fig. 43). When these raise cover doors are closed, an effective seal is established. This seal has been made use of in establishing a system of air splits. The ascending air column is not permitted to pass through the entire mine, but is withdrawn on the levels below the fire breaks and directed into the main return. Fresh air is distributed on the level above the break, thus assuring that each section of the mine is served by a split with reasonably

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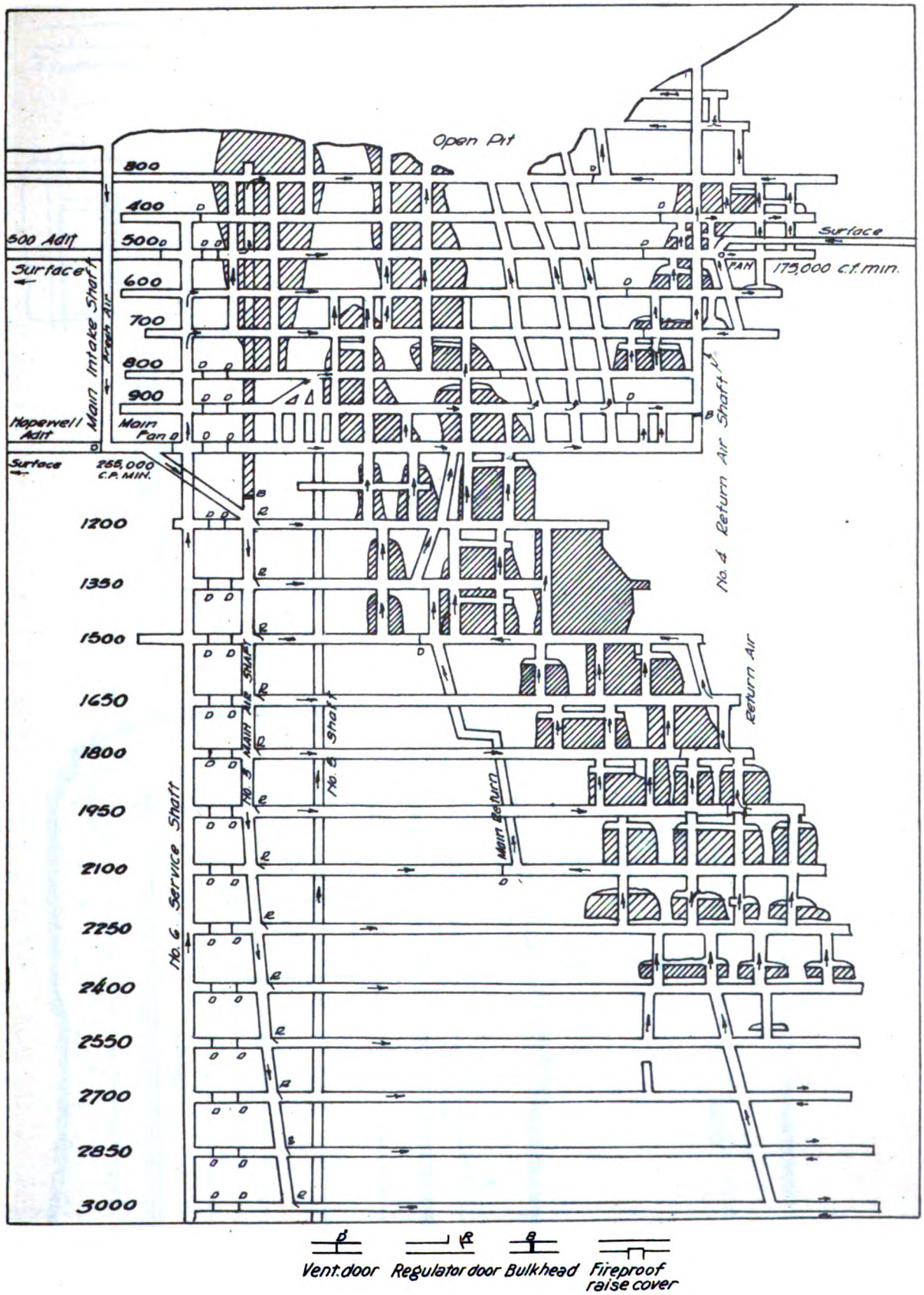

Figure 42.- Diagrammatic section, general ventilation system

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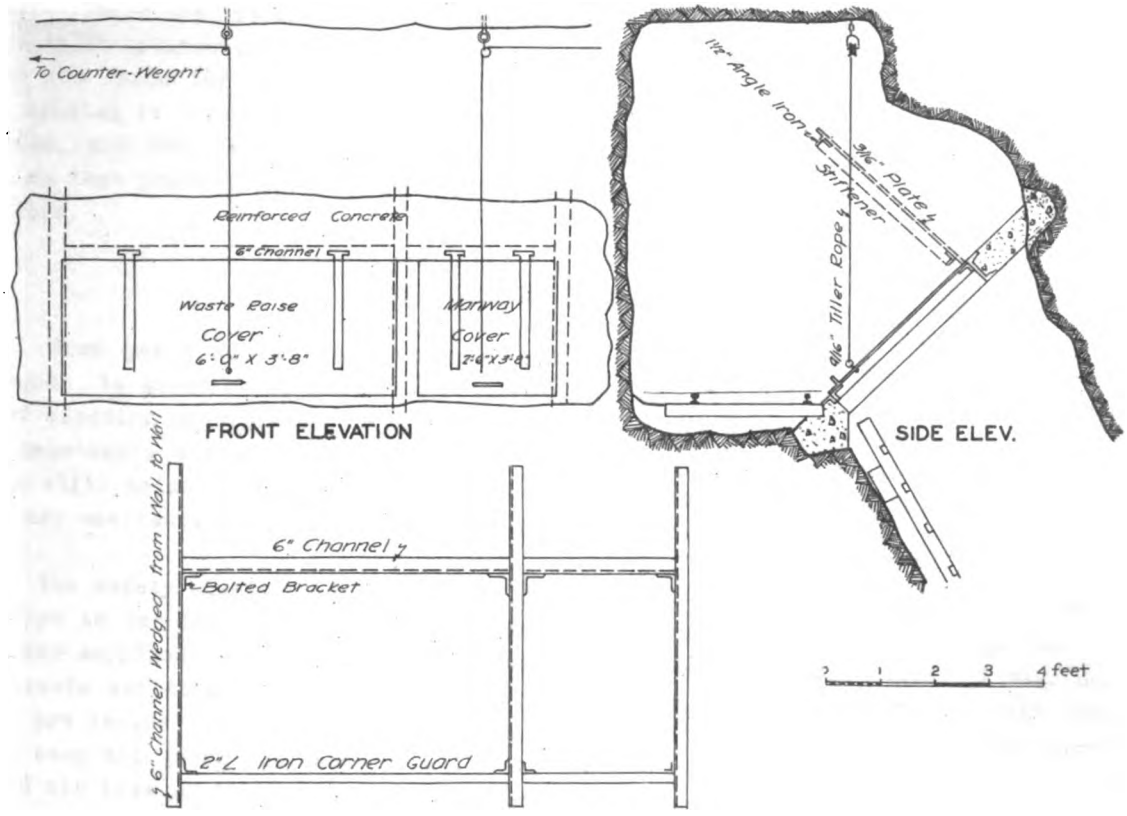

Figure 43.- Raise cover, showing doors and supports

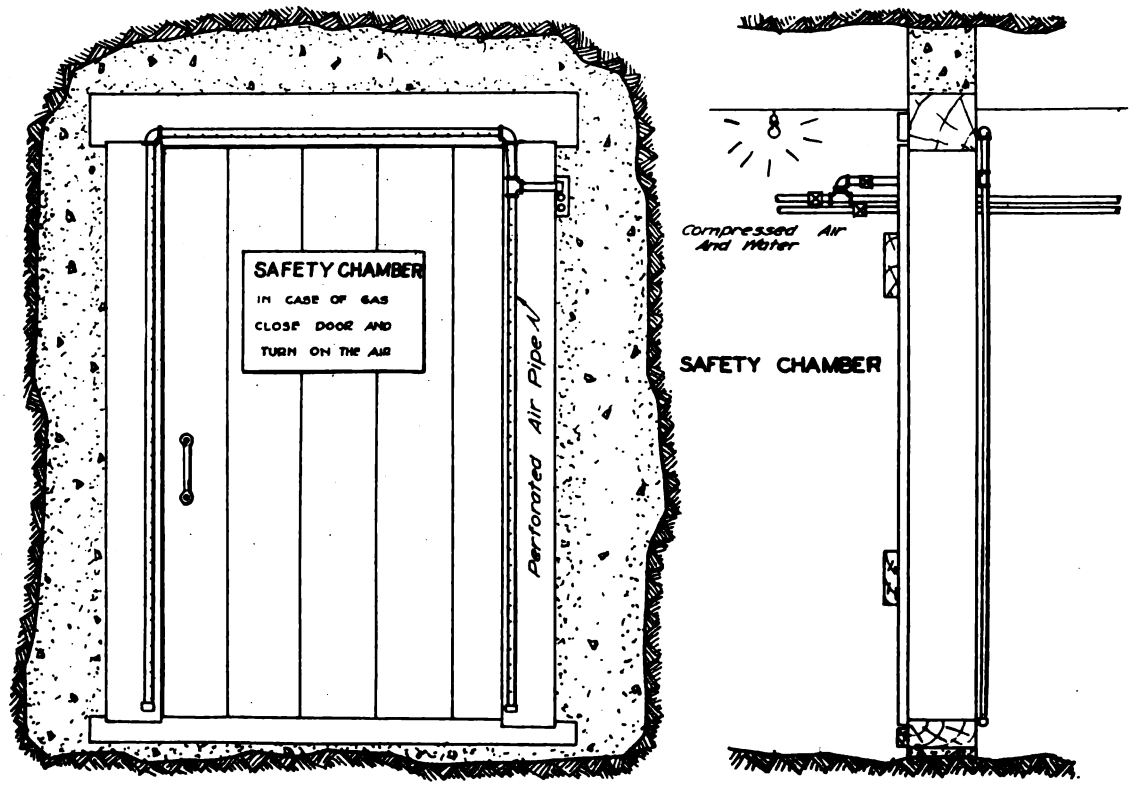

Figure 44.- Safety-chamber door

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fresh air. There are, of course, isolated sections where small volumes of vitiated air pass from one split upwards into another. This is unpreventable in the course of mining, and has not been the cause for complaint because it has always been possible to withdraw such air without sending it through other active workings. Seals of the nature described exist on the 1950, 1500, and 1200 levels. In case of a fire, it is expected that the smoke can be controlled so that production will have to be stopped only on the levels immediately affected by the fire.

Safety Chambers

Some years ago safety chambers were of vital necessity for the safety of men underground. In more recent years their importance has not been so pronounced. The present method of blasting on signals, and the greater control now exercised over the air currents, gives a reasonable assurance of a safe exit. However, as an additional safety precaution, there are still maintained on each level one or more safety chambers into which men may retreat in any emergency and feel safe as long as the compressed-air lines are intact.

The safety chambers are located in dead ends of drifts. A standard-type mine door is installed as far from the face as space outside the chamber will permit (fig. 44). The chambers are supplied with electric lights, water, and compressed air lines. A sign on the door instructs men to close the door and turn on the air. Valves controlling both the air and water are inside the chamber. The escaping compressed air builds up sufficient pressure within to keep all gases out under normal conditions. As an additional precaution a 1-inch perforated air line is fastened on the outside of the door opening. Signs at various points on the level indicate the direction to the nearest safety chamber so that men can readily find them.

Auxiliary Ventilation Equipment

Auxiliary equipment is used to ventilate dead ends and development headings. This equipment consists of electrically-driven fans and compressed-air blowers of the injector type, connected to metal pipe or flexible tubing. The injector blowers are used only on very temporary jobs and where the air has not over 300 feet to travel. They are also used as line boosters, operating in series with an electrically-driven fan. Metal pipe is now only used in exhaust lines, and where heavy electric blasting is done.

Mine Fire Methods

The fire hazards in the United Verde mine are, with perhaps one exception, no different from those in any other mine. That exception is the heavy sulphide ore, the dust of which is known to cause explosions when permitted to accumulate. An explosion of this kind is very apt to cause trouble in a timbered stope.

Much is done to elininate fire hazards and to prevent fires. Fire patrolmen are employed who inspect not only the active workings, but spend considerable time in temporarily or permanently abandoned workings.

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The gradual elimination of trolley wires is considered a move in the right direction toward the prevention of underground fires. Trolley motors are still in use on six levels, but 15 levels have storage-battery equipment.

Carbon tetrachloride fire extinguishers are located at all charging panels, and near all other types of electric equipment. Two and one-half gallon Foamite extinguishers have been placed on various levels close to heavily timbered areas. It is not expected that these extinguishers will be used to fight fires, but rather to put out incipient flames which might otherwise become serious fires. Ten fires of various kinds were extinguished between June and September, 1929, with hand-operated equipment. In addition to the above, the mine is equipped with a 5-inch water column to supply water for fires only. The compressed-air and water pipes on each level are connected to the column on the shaft stations, with valves so arranged that there is no interference in the flow of air or water under normal operating conditions. A blue print showing all connections and valve arrangements is kept under glass at each level station. Hose connections have been put in all water lines at 100-foot intervals where these pass through timbered areas. Special high-pressure hose in 50-foot lengths is kept in racks at strategic points on the levels. A 100,000 gallon capacity storage sump is located on the 1000 level. This is kept full of water from the main drainage system. A triplex pump, with a capacity of 159 gallons per minute is capable of pumping the water out of this sump to the surface storage or direct to any level desired. This system of utilizing the mine drainage water for emergency purposes has been found desirable because of the seasonal shortage of surface water.

Mine-Rescue Crews and Equipment

An active, trained rescue crew of 40 men is maintained. These men receive training once every two months in the use of oxygen breathing apparatus and gas masks.

Rescue equipment consists of 10 two-hour Gibbs breathing apparatus, 5 two-hour McCaa, and 3 one-half-hour McCaa breathing apparatus, and 20 Burrell's All Service Gas Masks. Two high-pressure oxygen pumps, one of which is mounted on a mine truck, one low-pressure fan, also mounted, and two fire-tool trucks, complete this equipment. One portable telephone is also used with this equipment.

Mine Safety

The safety work at the United Verde mine consists of inspections, education, safety advertising, and supplying approved safety equipment.

Every official is a safety inspector so that there is a constant inspection rather than only periodic inspection by one man. Inspections are nevertheless carried out periodically by the safety engineer. Safety committees chosen from among the miners and shop men make weekly inspections and report to their shift boss or foreman.

The principal mine accident hazards may be summarized as follows:

1. Fall of ground from roof and walls.
2. Flying of sulphide when collaring a drill hole or breaking rock with hammers.
3. Mine haulage.

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4. Handling rock. This frequently causes severe cuts on hand and arms.
5. Sulphide gas generated when blasting in heavy sulphides.
6. Falls of men and material.

Safety education consists of lunch-hour talks with the men, bulletins, and photographs of the right and wrong way of doing work. Safety advertising consists of bulletins, pictures, blackboard announcements, and contests.

Safety equipment sold by the company at cost consists of "hard-boiled" hats, gloves, and respirators. Goggles are issued when men are hired. In addition to this equipment, all men are required to wear hard-toed shoes, which are sold by Jerome merchants. Safety belts and chains are issued upon request from the shift boss only.

The hard, brittle character of the ore makes it absolutely necessary that goggles be worn when starting a hole or when breaking rock with a hammer. Hard and fast rules to this effect have been made and must be observed on penalty of a lay-off. The hard-boiled hats and safety shoes which are required as a part of the miner's equipment have unquestionably prevented many serious accidents and, no doubt, a number of fatalities.

Foremen and shift bosses are held strictly responsible for the safety of the men under their supervision. They must see that the tools they use are in good repair, that the working place is safe, that the men observe all safety rules, and otherwise conduct themselves in the safest manner possible. The shift boss is expected to instruct the new man in his work, the hazards connected with it, and how best to eliminate or avoid them. The old employee is constantly reminded of the fact that safety should receive his first consideration on any work that may be assigned to him.

Certain rules are constantly stressed and impressed on old and new employees alike; violation of them means a lay-off or discharge from service.

A safety contest in the form of a horse race was started in July of 1929 and ended on December 31. Each shift boss and his crew was represented by a race horse. The horses were advanced several times a week and placed in their relatively correct position once each month. The position of any horse was dependent upon the frequency rate of lost-time accidents per 10,000 shifts. The crews without accidents were, of course, in the lead at all times. Seven horses tied for first place. A prize was given to each member of the winning crews. Another method used to stimulate interest in safety is that of giving out cigars at the end of each month to those crews which have gone the full month without a lost-time accident. The cigars are wrapped in foil and have printed on the wrapper:

In appreciation of your efforts
in Safety and Accident Prevention.
United Verde Copper Company.

The shift bosses too are rewarded for an excellent safety record. A monthly safety bonus is paid to them. An honor roll is posted each month, containing the names of those shift bosses who have gone three consecutive months or more without a lost-time accident to any of their men. Shift bosses having 10,000 man shifts to their credit without a lost-time accident receive a special award.

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While there is some expense connected with this method of promoting safety, records show that it is very effective. It is believed that once the value of safety is firmly imbedded in the minds of the men, much in the line of contests and awards can be eliminated.

Figure 45 shows a comparison of lost-time accidents in 1928 and 1929.

MINE DRAINAGE

The Hopewell tunnel on the 1000 level, which is the ore-haulage level for the mine, is also the mine drainage tunnel. All mine water above this horizon is collected on the various levels and, by means of ditches and diamond-drill holes, passed from level to level until it reaches the Hopewell tunnel, through which it flows to the surface. The water collected below the 1000 level is lifted to the Hopewell tunnel by two pumps. Both are 6½ by 12 inch quintuplex pumps, with capacities of 500 gallons per minute each, lifting 1,000 feet vertically. They are geared to 150-horsepower, alternating-current motors. The main pump column is an 8-inch, lead-lined pipe, hung in the pipe compartment of No. 6 shaft.

The drainage water on the various levels is ditched to the main haulage crosscut and directed to within several hundred feet of No. 6 shaft, and there passed through a diamond-drill drainage hole downward to the next level. Thus the water between the 1000 and 1950 levels drains into the 1950-level pump sump; below the 1950 level it drains into th 3000-level sump. The diamond-drill drain holes are 2½ or 3 inches in diameter.

The pump on the 3000 level is operated about three hours per day and lifts its load to the 1950-level sump. The pump on the 1950 level runs about five and a half hours out of the 24, and discharges its water into the Hopewell tunnel drainage ditch.

Approximately 180,000 gallons of water flows through the Hopewell tunnel daily. On the surface this water is utilized in leaching operations on low-grade ores.

MINE ORGANIZATION

The administrative organization at the mine is shown in Figure 46.

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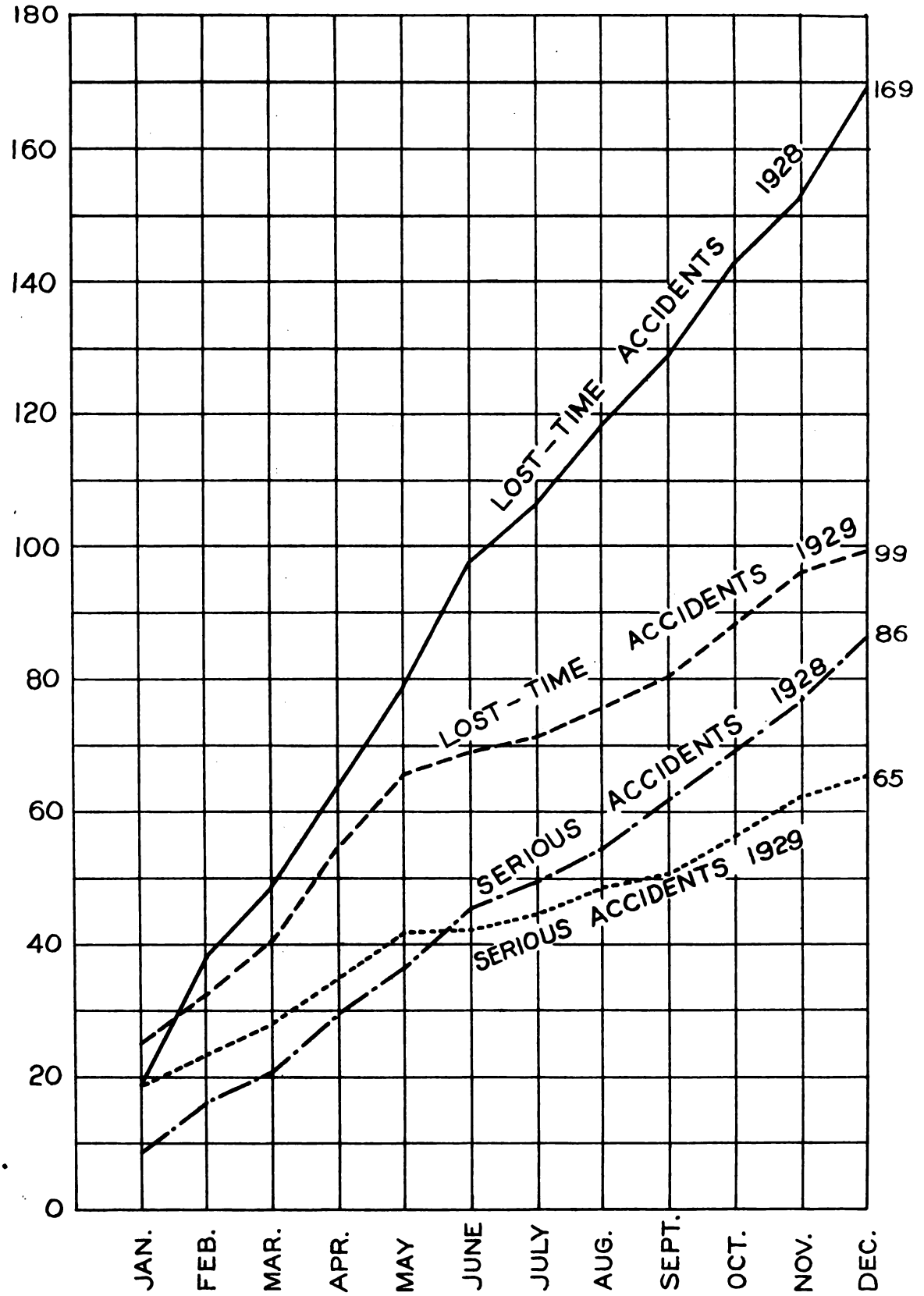

Figure 45.- Accident frequency curves, 1928 and 1929

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	Development, hrs. per ton	Mining, hours per ton			
		Cut-and-fill	Square-set	Top slice	Shrinkage
Breaking	[1]0.157	0.135	0.383	0.383	0.248
Mucking	.119	.547	.872	.651	.559
Timbering and fill	- -	.466	.606	.837	.352
Haulage and hoisting.	- -	.238	.238	.234	.260
Supervision	.006	.122	.122	.120	.134
General	- -	1.037	1.039	1.020	1.136
General	- -	1.037	1.039	1.020	1.136
Total	0.282	2.544	3.260	3.245	2.689

1 Development figures show hours on development against total stope tonnage; 0.157 hours per ton is for both breaking and timbering.

	Development[1]	Cut-and-fill	Square-set	Top slice	Shrinkage
Average tons per shift	28.33	3.14	2.45	2.47	2.97
Labor, per cent total time	9.19	53.01	28.08	8.32	1.40
Powder, pounds per ton[2]	.45	.55	.60	.71	.71
Timber, board feet per ton	.83	6.95	12.52	9.91	2.36
per ton	.83	6.95	12.52	9.91	2.36

1 Development figures are computed against total stope tonnage.

2 Principal powder used is 1⅛ by 8 inch, 50 per cent gelatin.

Average tons per man-shift on surface charged to underground operations is 74.95.

	Air compression	Hoisting	Pumping	Ventilation	Lighting
Power kw.h. per ton	10.15	2.35	0.26	3.07	0.61

Development details in units of labor, power, and timber.
Average drift and crosscut is 6 feet wide by 8 feet high.
Average raise is 6 by 11 feet with one timbered compartment.
The rock is generally hard and firm.

Labor	Drifts and crosscuts, hours per foot	Raises, hours per foot	Total
Breaking and timbering	3.50	9.15	4.75
Mucking	4.17	1.56	3.59
Supervision	.19	.19	.19
Supervision	.19	.19	.19
Total	7.86	10.90	8.53
Feet advance per 8-hour shift	1.02	.73	.94
Powder, pounds per foot[1]	13.56	14.36	13.74
Timber, board feet per foot	4.57	97.02	25.03

1 Principal powder used is 1⅛ by 8 inch, 50 per cent gelatin.

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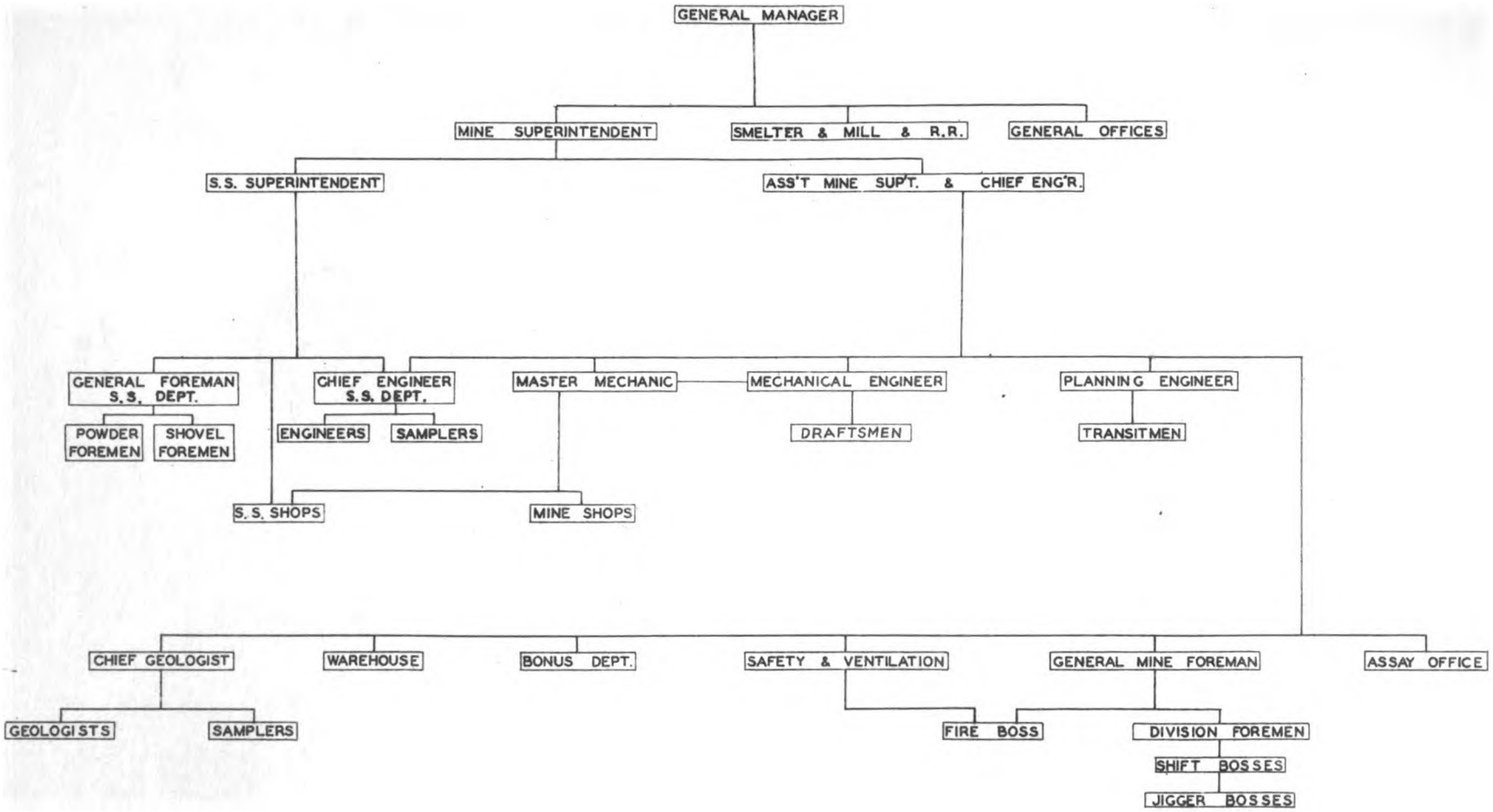

Figure 46.-Organization chart

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